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New York, March, 1912.

Editorial:
Rail Quality and Rail Duty ... 123
Lead and Zinc Tariff ... 124
"Form" Value and the Metallurgical Industry ... 125
Boston Meeting of the American Electrochemical Society ... 126
Fixiation of Atmospheric Nitrogen in South Carolina ... 126
The Patent Reform Problem and the Inventors' Guild ... 126
The Western Metallurgical Field ... 127

Correspondence:
The Position of the Induction Furnace. By T. D. Robertson ... 128
Discussion at San Francisco on Alaska Coal Mines. By R. W. Raymond and Jos. W. Richards ... 128
Refractories. By Francis A. J. FitzGerald ... 130
Concentration by Flotation of Dry Ore Pulp. By H. C. Parmelee ... 132
Electrolytic Determination of Lead. By Elmer List ... 135
Calculation of Furnace Charges.—III. By Regis Chauvenet ... 135

Symposium on Research ... 140
With Abstracts of Papers by F. A. Lidbury, A. M. Comey, P. McN. Beanie, and Chas. F. Lindsay.
Effect of Electrical Current on Cyanide Plating Solutions and a Simple Way for Their Preparation. By M. G. Weber ... 152
The Important Factors in Blast-Roasting. By H. B. Pulsifer ... 153
The Flare of Furnace Walls. By C. Hering ... 159
Influence of Increased Temperature on the Strength of Certain Metals and Alloys ... 160

New York Meeting of the American Institute of Mining Engineers ... 161
Abstracts of papers by J. G. Klugh, W. S. Landis, F. A. Vogel, I. W. Richards, N. V. Hansell, H. A. Wentworth, C. Q. Payne, H. H. Emrich, H. O. Hoffman, W. Wanjukow, and F. G. Cottrell.
Notes on Chemistry and Metallurgy in Great Britain ... 174
Synopsis of Metallurgical and Chemical Literature ... 175
Recent Metallurgical Patents ... 176
Atomized Metals ... 180
The Use and Care of Crucibles in Foundry Practice ... 182
Latest Progress in Optical Pyrometry. By Arthur Lenk ... 183
Purity of a Requisite for Silver Electroplating Anodes. By J. L. R. Brown ... 184
Illuminated-Dial Pyrometer ... 186
The Iron and Steel Market ... 186
The Non-Ferrous Metal Market ... 187
Personal ... 187
Obituary ... 188
Notes ... 188
Digest of Electrochemical and U. S. Patents ... 188
Book Reviews ... 189

Rail Quality and Rail Duty.

No doubt it is fully realized by the representatives of the rail mills and the prominent railroad men who are now discussing the question of making steel rails better that the service which rails are expected to perform has increased greatly and that the problem is simply the engineering one of finding the rail which at the least expense will meet the requirements. The general public in the iron and steel trade perhaps does not fully realize how greatly the duty upon rails has increased, or how it chances that at this particular time the problem should require so much attention. The case, however, is a very simple one. From the time steam railroads were first built the duty has been constantly increasing through increases in speed, in pressure of wheel upon rail, and in density of traffic. Precisely in what measure each of these influences tends to wear rails out faster cannot be stated mathematically, but obviously they all do tend in the same direction.

In 1850 a 50,000-pound locomotive was considered heavy. Such locomotives generally had eight wheels, so that the heaviest weights per wheel generally ranged from 5000 to 6000 lb. Last year about a score of locomotives were ordered weighing more than 400,000 lb., involving an average weight per wheel of about 30,000 lb. Many were ordered between 300,000 and 400,000 lb., entailing an average weight per wheel of about 25,000 lb., while weights per wheel of locomotives ordered between 200,000 and 300,000 lb. weight averaged above 20,000 lb. per wheel. To a considerable extent it may be taken that in the matter of wheel pressures it is not the average, but the greatest wheel pressure which determines whether or not the rail is equal to the load, and it may be taken that wheel pressures have been multiplied by five in the past sixty years.

As to speeds we have no precise data, but it will probably be generally admitted that the speed of railroad trains has doubled in less than half a century.

Poor's railroad statistics give for many years the total track mileage in the country and the total ton-mileage of the country. From these we compile the average ton-mileage per mile of track: 330,000 in 1885, 380,000 in 1890, 377,000 in 1895, 550,000 in 1900, 630,000 in 1905, and 750,000 in 1910. It will be observed that in the fifteen years from 1895 to 1910 the average density of traffic has precisely doubled. This is the average, and upon the most traveled track there is reason to believe the density has more than doubled.

A multiplying of weight on drivers by five in sixty years represents an annual increment of 2.7 per cent; a doubling of speed in forty years represents an annual increment of 1.7 per cent, and a doubling in density of traffic in fifteen years represents an annual increment of 4.7 per cent. We have no reason to assume that these factors are of equal moment in destroying rails, but there is no harm in observing that the continued product of the three factors represents an annual

increment of 9.5 per cent, which is a doubling in less than eight years.

It would be far from surprising, then, to find rails wearing out twice as fast as they did five years ago, while it would be surprising not to find them wearing out twice as fast as ten years, merely by reason of the increased duty expressed in the above figures.

It may be wondered why the problem of rail wear and rail breakage has come up with such prominence in the past few years, when the increase in service required has been so continuous, from the very beginning of railroading. The reply readily occurs. At first iron rails were used, and for many years they proved adequate to the service required, their weight being increased as necessary. There did come a time in the history of the iron rail, however, when it was regarded as very unsatisfactory, and many efforts were made to improve it, particularly by the manufacture of an iron rail with a steel head. Suddenly there came the Bessemer steel rail, which was by nature vastly superior to the iron rail. A sudden jump was made, and the improvement was amply sufficient to take care of the continued increment in service required for many years. The problem was laid on the shelf for a generation. In recent years it has become prominent. A few years ago railroad engineers started to increase the carbon rapidly in order to secure greater durability, and the increases were pushed to such an extent that breakages have now become rather frequent.

It is not, as in most engineering problems, a case of choice of evils. A certain percentage of breakage cannot be tolerated in order to provide greater wearing quality. The rails must be made safe first and durable afterwards.

A very distorted impression obtains in many quarters as to the average life of a steel rail. Much publicity has lately been given to statements that the rail mills formerly guaranteed five years' life to their rails, but will no longer do so. Such a guarantee is really an absurdity, for on some stretches of track the duty is many times as severe as upon other stretches. Over a limited mileage rails do wear out in a very few years, but on an average they last a very long time. For strictly replacement purposes the railroads have hardly yet reached an annual consumption of 1,000,000 tons a year, yet they have 360,000 miles of track. Laid in 90-lb. rails this would be 50,000,000 tons and in 80-lb. rails 45,000,000 tons. The average may be less than 40 lb., but the evidence is that the average life of a rail is nearer forty years than five years, which, as an average, is not so bad after all.

Unquestionably the improvement of the rail involves an increase in the cost. It is not, as carelessly assumed often, that the cost of all rails must be increased. When rails wear out very slowly the annual fund required against ultimate replacement is much smaller than the interest charge, but when rails wear out in a very few years the replacement expense is much greater than the interest expense. What is needed is a wide range of rail quality, at varying prices. All the rails should be made safe, which can be done by sacrificing a portion of the extreme of durability which has lately been sought by the unsafe method of increasing the carbon without heat treatment or other palliative. Then those which are subjected to heavy duty should be improved in wearing quality by methods which rail makers can readily develop, when the railroad engineers place themselves in position to know how much they can afford to pay, in price per yard of rail, for the varying degrees of heavy service.

It is probable that the chemical specification is of much less account than the physical structure—the rail must be made physically sound, free from slag enclosures, etc. The great success of titanium treatment points in this direction as the titanium acts chiefly as a purifier and only in special cases as an alloying element. The success of electric furnace treatment would also rest to a large extent on an analogous basis; since the resting of the steel in a non-oxidizing atmosphere in the electric furnace would permit thorough removal of all gases, slag particles, etc. In this respect the results obtained this winter on Northwestern lines with 5000 tons of electric steel rails (made from steel refined in the Heroult furnace) are significant, as there was not a single breakage of these rails, while there were some 200 breaks of Bessemer and open-hearth rails of the same weight and under the same kind of service on the lines in question. This seems to augur well for the ultimate success of the electric steel rail.

The Lead and Zinc Tariff.

Congress has stirred the metallurgical industry by a proposal to admit zinc ores to the United States free of duty, and to reduce the tariff on lead ores and products. The usual political hue and cry is accompanying the consideration of a subject which should receive unbiased treatment at the hands of disinterested parties, and it is more than likely that the true merits of both sides of the question will be totally obscured by the prominent part which politics will play. For this reason the feeling exists that, whatever action Congress finally may take, it will hardly be the result of a scientific investigation of the industrial condition, but rather of political trading to secure local benefits.

The Underwood bill, with its proposed tariff changes, has aroused almost unanimous opposition among ore producers and smelters alike. Under the leadership of the American Mining Congress the ore producers have been called upon to protect their mining interests from an influx of Mexican and Canadian ore, while the smelting interests have appeared on protesting committees in various parts of the country. In Salt Lake City a committee of mining men has endeavored to gather data and statistics on which they may base an intelligent opposition to the proposed reduction. In Denver, however, where the subcommittee of the Chamber of Commerce was composed of both smelting and mining men, the effort to adopt resolutions opposed to the reduction was vigorously antagonized and finally laid on the table. In Spokane, Washington, where mining men are directly interested in Canadian zinc and lead deposits across the international boundary, the sentiment favors the free entry of these ores.

The principal argument against the tariff reduction is that Canadian and Mexican ores would have an undue advantage over domestic production and consequently would glut the local market to the exclusion of the domestic product. On zinc ores containing 25 per cent or more zinc, the tariff is 1 cent per pound of zinc contained. On lead ores the tariff is 1½ cents per pound of lead in the ore. Again, it is claimed that the

cheap labor available in Mexico very favorably affects the cost of production in that country, and that Mexican zinc ore would have this additional advantage.

On the other hand, it is argued by a producer of zinc concentrates that, despite the fact that the past year has witnessed a wide margin between ore and spelter, a condition inuring greatly to the benefit of the smelter, there has been no corresponding benefit to the producer. In fact, the base price paid for concentrates has been reduced, and the producer has not been given as much benefit from market variations as formerly. Hence, although the spelter market has been in good condition, with a large European demand and steady home consumption, the producer has not reaped his share of the reward. The argument regarding cheap Mexican labor also is questioned. Writers on labor conditions in Mexico usually have agreed that the labor is cheap in every sense of the word, and that for responsible and reliable work skilled labor must be employed. In Canada the labor conditions are more nearly comparable to our own, but the ores are complex and require careful dressing to prepare them for market. The cost of transportation to domestic smelting centers also would be very great, and it is doubtful if much zinc ore could be shipped from Canada at a profit.

It is apparent from this brief view of the situation, that opinions differ slightly. But whatever the result of the political agitation may be, it is evident that there is some ground for feeling that more equitable conditions should exist between producer and smelter wherby the former will share the benefit accruing from any improvement in the industry.

"Form-Value" and the Metallurgical Industry.

Economists say that value is created when any natural or artificial product is turned into some form useful to the community and that value is of three kinds: "time-value," "place-value," and "form-value." For instance, ice on the upper Hudson in winter has little value there. When brought to New York City it has "place-value" and in the summer it has an increment of value—"time-value." The 80 calories per gram of its latent heat of fusion abstracted from flavored milk turn it into ice cream and render it of "form-value." In each case there is a sort of commercial potential existing between consumer and the commodity and the owner of that commodity exacts his price for the use of the thermal potential of 80 calories.

Turning from this chilly subject to the mineral industry, one cannot fail to notice the marked increase of value given the mineral products of the United States by the various processes that give them "form-value." Take, for example, two tons of iron ore turned into one ton of steel. According to the degree of refining this steel sells at a price of from 1 cent to 10 cents per pound in the semi-finished form. When placed in an automobile it sells with the other materials for $1.00 per pound In the form of fine cutlery it sells for $5 to $10 per pound. In all of these forms it has acquired its "form-value" or commercial potential by the use of chemical and metallurgical processes that consummate the production of a refined metal. Without this special technical ability the mechanical industry could not exist. Indeed it is quite clear that the metallurgical and chemical industries stand in the same relation to the general manufacturing community as does the agricultural industry stand to the entire community. They each furnish the essential of the depending communities' existence.

In a most intense aspect, the electrochemical industry especially exhibits great power to impart "form-value" to natural products. This is natural for the use of such a powerful agent as the electric current should impress increased value on the raw material operated upon.

In the electrolysis of brine producing chlorine gas and caustic soda, 1 ton of salt worth on an average of $5 yields products worth six or seven times that amount. The graphitizing of impure coal makes a product a hundred-fold more valuable. The same is true of the calcium carbide furnace, the carborundum furnace, and the ferro-alloys industries. In its extreme, almost ludicrous, instance we see the nitrification of air. Air has great utility but usually no value, for the supply is practically unlimited. Here the ratio of natural value to "form-value" has reached the limiting case, amounting to infinity.

Another example is the electrolytic refining of crude copper bullion carrying precious metals, where the "form-value" is great but assumes a cryptic aspect. The value to the community is enormous, for without its operation the electrical industry would be throttled for lack of high-conductivity wire. The price paid for this service is slight because it was not possible for the innovators to gain any monopolistic rights by patent protection. But nevertheless the enhanced "form-value" is present and society little appreciates what they owe to such pioneers as Edward Balbach and William Thum, Sr.

The total value of the mineral industry in the United States for the calendar year 1910 amounted to $2,003,744,869 according to the report of the United States Geological Survey. This is always given as "spot" value, *i. e.*, at that point of primary production. In general, there is but little "form-value" or "place-value" in this figure, stupendous as it is. If we consider the great increases in value probable by the working up of these crude products into more valuable forms, having in mind the fact that the ratio between value of finished product to crude product always is large and in one instance amounts to infinity, we arrive at a conclusion that the increment to our national wealth seen tangibly in our cities and factories is in greatest measure due to our production of mineral wealth. We have as a necessary corollary that our measure of national prosperity depends on the efficiency of the chemical and metallurgical and mechanical processes for the production of these essentials of modern civilization. All those who improve such processes and thereby heighten the prosperity deserve well of their fellows and of society. The technical schools and the universities, the engineering societies and the technical press, all combine to bring about increased general efficiency in the industrial world. It is a favorable sign that such endeavors are heartily received in times of apparent depression such as have been prevalent recently. With the great advance in the knowledge of physical chemistry, of all science, and of sound business principles in every manufacturing line and the wide diffusion of such intelligence, it is only fair to assume that the creation of "form-value" will be easier and more efficient in the future. It must be a pleasure and a privilege for each of us individuals to assist in such a great task.

Boston Meeting of the American Electrochemical Society.

The date of the annual meeting of the American Electrochemical Society has now been fixed. The meeting will be held on Thursday, Friday and Saturday, April 18 to 20, in Boston, Mass.

One of the features of the professional sessions will be a symposium on conductivity in its broadest aspects.

A fuller program will be published in our next issue.

Fixation of Atmospheric Nitrogen in Nitrolee, South Carolina.

In our Volume VIII, page 555 (October, 1910,) we referred to the decision of the Southern Power Company to enter, through a subsidiary company, the field of fixation of atmospheric nitrogen for the production of nitrate fertilizers for which there is a good market in the South.

The name of this subsidiary company is the Southern Electrochemical Co. and its plant at Nitrolee, S. C., has recently started operation. The plant has a capacity of 4000 h.p., the power being supplied from the two large waterpower plants of the Southern Power Co. at Great Forks and Rocky Creek.

The output is estimated as 8 tons of calcium nitrate per day of 24 hours. Calcium nitrate is in successful use in Europe as a fertilizer in competition with Chili saltpeter.

As has already been noticed, the Pauling process is being used in the South Carolina plant. It is already in successful commercial use in Tyrol and in France and has been described in some detail repeatedly in our former volumes, especially in our Volume VII, page 430, where the plant at Innsbruck, Tyrol, is described, and in our Volume IX, page 102 and page 196, where the plant at La Roche de Rame, near Briançon, Hautes-Alpes, France, is described.

The Patent Reform Problem and the Inventors' Guild.

In connection with the discussion of the patent reform problem in our January issue (page 8), we are glad to report the adoption of a resolution, addressed to President Taft, by the Inventors' Guild, which was particularly qualified to do so by the very high professional standing of its members. The resolution has been forwarded to Washington immediately after its adoption by the Inventors' Guild through its president, Mr. Ralph D. Mershon.

The strong point of the resolution is the request for the appointment of a joint Congressional committee, and it is expected that other technical and industrial organizations will join the Inventors' Guild in urging the appointment of this committee.

We herewith print this resolution in full.

"Whereas: The Constitution of the United States provides: 'The Congress shall have power to promote the progress of science and useful arts, by securing for limited times, to inventors, the exclusive right to their respective discoveries.'

"And whereas: This constitutional provision was intended to obtain for the benefit of the nation the publication of every new and useful invention in such full, clear, concise and exact terms as to enable any person skilled in the art or science to which it pertains to make, construct, compound or use the invention, after the limited time for which the exclusive right is secured to the inventor by patent; and thereby to secure for the nation the great benefit which, all experience shows, results to a nation from publishing inventions, in contradistinction to following a policy which would tend to encourage trade secrets, monopoly and trade combinations, which minimize the value of inventions to the nation;

"And whereas: A patent is in effect a contract between the government and the inventor by which the government, in consideration of the right to publish the invention for the benefit of the nation, agrees that in return for his satisfactory disclosures of his new and useful invention under reasonable conditions, to be determined by the government, it will secure the inventor for a limited time in the exclusive right to his new and useful invention.

"And whereas: An inventor, after having performed his part of the contract by having made proper disclosure of a new and useful invention to the United States government officials, is frequently subjected to unreasonable delay, expense and injustice before obtaining his patent, and after having obtained his patent is not equitably secured in his exclusive right as the Constitution intended that he should be secured in return for his disclosure in good faith of his new and useful invention; and as a consequence of this unfair treatment of inventor patentees, the United States is not obtaining, in the degree that it should, the national benefit of the best inventive work of its many able inventors.

"And whereas: The United States patent system has been evolved to its present condition without proper consideration of the rights of the nation, and of the inventors, who are the two real parties at interest, but on the contrary has been developed to its present condition almost entirely as the result of suggestions from persons who do not occupy the position of one of the parties to the contract which every patent represents; and who do not suffer damage from the delays, complications, injustice and expense characteristic of the United States patent system and the United States courts which hear patent causes; said damage being borne principally but indirectly by the nation and to a lesser degree, but directly, by the inventor-patentees.

"And whereas: It is a well-known fact that modern trade combinations tend strongly toward constancy of processes and products, and by their very nature are opposed to new processes and new products originated by independent inventors, and hence tend to restrain competition in the development and sale of patents and patent rights; and consequently tend to discourage independent inventive thought, to the great detriment of the nation, and with injustice to inventors whom the Constitution especially intended to encourage and protect in their rights.

"And whereas: Under existing methods of trying patent causes, an inventor-patentee of average means could not, at his own expense, carry to a conclusion an average patent litigation against a wealthy opponent, and therefore a few wealthy concerns usually acquire nearly all important patents in their field, to the great damage of the nation because of the restraint of competition and because of the resulting tendency of such inventors to seek protection for their inventions by trade secrets or else to cease inventive work.

"And whereas: Efficient protection by patent of new and useful inventions would offer to the average American manufacturer one of the best methods of meeting foreign competition and would, in addition, improve quality, reduce first cost and stimulate fair competition with resulting benefit to the entire nation.

"Resolved: The Inventors' Guild composed exclusively of independent and experienced inventor-patentees, does hereby respectfully ask the attention of the President of the United States to the urgent need of reforms in the patent office, and also in the courts which hear and decide patent causes; and hereby requests the President to recommend to Congress the advisability of appointing a committee to confer with experienced and representative inventors with the object of promptly accomplishing such reforms as will result in more effectively carrying out the intention of the Constitution; and to supplement such recommendations by such executive action as in his judgment seems likely to assist in accomplishing the needed reforms."

The Western Metallurgical Field.

Cyanidation at Cripple Creek.

The excellent work done last year at the Victor mill of the Portland Gold Mining Company, is attracting very favorable attention. Not since the successful operation of the Stratton's Independence mill has such progress been made in the treatment of low grade ore. Indeed the feature of the work at Stratton's was the utilization of old dump ore already mined, but at the Portland the chief merit lies in the fact that low grade mine ore extracted in the course of development work and in the mining of high grade ore is being made to pay a good profit. One feature of the metallurgy still remains undisclosed, viz., the details of the Portland cyanide process which embodies the chemical treatment to which the ore is subjected.

In January, 1911, the mill treated 8052 tons of ore, but the capacity of the plant was increased during the year so that in December, 1911, the total ore treated amounted to 12,285 tons. The gross value of the ore treated during the year was $3.51 per ton, but in December the value of the ore was only $2.80 per ton; but even on this lower grade ore the mill made a recovery of 80.5 per cent at a cost of only $1.1317. The extraction for year amounted to 81.4 per cent, netting a profit of $128,738.72 on rock which formerly was worthless to the company and thrown on the dump. As the total cost of construction of the mill, as officially announced, was only $338,580.96, it will be seen that the profits from another year's operations combined with those of the year and a half passed, will wipe out the cost of construction; and as there is a sufficient tonnage of low grade ore available to keep the mill running for a number of years to come, the profits will be a very important item to the company.

Among the improvements made at the mill since the beginning of this year, is the installation of a large Portland filter of the revolving drum type. The former equipment filters have the following dimensions: diameter, 12 ft.; face of drum, 9 ft.; capacity, about 45 tons of pulp in 24 hours. The new large filter is 14 ft. in diameter, with a 12-ft. face, and has a capacity of 90 tons of pulp in 24 hours. In addition to the larger capacity, there are advantages in the deeper submergence of the filter in the pulp, and better opportunity for washing the cake free of cyanide solution and dissolved metals.

At the Colorado Springs mill of the same company, the old equipment for chlorination is being abandoned, and the mill is being converted into a modern cyanide plant. The preparation of the ore, including roasting, will be retained as before. It will then be ground in tube mills in cyanide solution and filtered in two 200-ton Merrill presses similar to those in use at the Homestake mine and other modern plants where filter pressing is adapted to the ore. Zinc dust precipitation also will be installed, making the plant conform to the most approved ideas in cyanidation of roasted ores.

At the company's mines in Victor there has been installed a complete equipment for life saving and fire fighting. This step has been taken as a precaution against serious accidents arising from foul mine gases and fires. A corps of twenty men have been instructed in the use of the apparatus by the United States mines rescue service.

As a result of drainage through the Roosevelt tunnel, the Portland mine is now free from water, and preparations are being made to prosecute development work in the lowest level.

Stope filling with mill tailing is another innovation which has been adopted during the past year. This has a double advantage in that it enables the company to take care of its old stopes at less expense than before, and at the same time leaves the surface of the property available for mining work.

Chemical Engineering Education.

The Colorado School of Mines is at present offering a new course leading to the degree of Ch.E. (chemical engineer), in addition to its regular courses in metal mining, coal mining, metallurgy and mining geology, for which the regular degree of E.M. is granted. In the freshman and sophomore years, the work is the same in all courses, but in the last two years of the courses special instruction is given in the line which the student has elected. In chemical engineering the special work of the last two years will include: Junior year: analytical and organic chemistry, stoichiometry, industrial inorganic chemistry, metallurgy of iron and steel, fuels, electricity, steam engines and boilers, mechanics of engineering. Senior year: electrochemistry, theoretical chemistry, physical chemistry, industrial organic chemistry, technology of the non-metals, electrometallurgy, assaying, geology of the non-metals, ceramic geology, rare metals and English. In addition to this required work, there are offered many elective studies from which the student may select such as are appropriate to his work, and as many as are needed to complete his required number of credits.

Chemical engineering is as yet confined mostly to the eastern states, but with the growth and development of the West there will undoubtedly be found a field for men educated in the principles of chemistry and engineering. Realizing this future need, our Western schools are anticipating the student's requirements, and are gradually offering him courses along modern lines.

Mason Valley Copper Smelter.

Since this plant was blown in January 6, one furnace has been in steady operation and is now treating about 435 tons of ore daily. This tonnage is being increased gradually and is expected to reach 600 tons when that quantity of ore is available. It is expected that the annual output of copper will reach 20,000 lb. when both furnaces are in steady operation, and judging from the inquiries being received from outside companies it is probable that the treatment of custom ore will become quite a factor in the operation of this plant.

Phosphate Deposits.

The increase in the acreage of phosphate land withdrawn from entry by the government, leads one to believe that perhaps the United States contains within its borders the greatest phosphate deposits in the world. During January 214,749 acres in the state of Idaho, believed to be underlain with phosphate rock, were withdrawn on the recommendation of the U. S. Geological Survey. This makes a total of 1,167,137 acres withdrawn in Idaho. In Wyoming, 1,266,668 acres are now withdrawn; in Utah, 107,745 acres; in Montana, 33,950 acres; and in Florida, 35,640 acres. This makes a grand total of 2,611,140 acres withdrawn in the United States. There is now need of the enactment of a law which will permit development of this great resource, which is coming more and more into use as the country grows.

Cement Statistics for 1911.

Preliminary estimates of the production of Portland cement in the United States during 1911, have been given out by the Geological Survey. They are based on the returns made by 70 per cent of the manufacturers, and are believed to be correct within 2 per cent. The figures for 1911 are very close to those of 1910, and although at present an increase of 1.7 per cent is indicated, this may be decreased when complete returns are in. Certainly any increase in 1911 over 1910 will not approach the increase of 17.7 per cent which occurred in 1910 as compared with 1909. The average factory price per barrel, not including containers was 86.7 cents in 1911 as compared with 89.1 cents in 1910, a decrease of 2.68 per cent. It is observed that the factory cost increases greatly in the West as compared with the East. The cost in the Lehigh district was 76.3 cents per barrel; in the mountain states, $1.156, and in the Pacific slope, $1.407 per barrel. The Lehigh district produced about 1/3 of the total output; Kansas, Oklahoma and Central Texas stood second, and the Pacific slope plants third in the list of producers.

Copper Company Reports.

The tenth quarterly report of the Nevada Consolidated Copper Company, for the quarter ending December, 1911, shows that the average output was 5,236,084 lb. per month. The production for December, 1911, was only 4,679,674 lb. due to the severe winter conditions which prevented normal operations. The grade of the ore treated in this quarter was 1.94 per cent copper.

The fifteenth quarterly report of the Utah Copper Company, covering the fourth quarter of 1911, shows an average monthly of production 8,939,416 lb. The total production for the year 1911 was 98,436,224 lb. In the quarter under consideration the total tonnage of ore treated was 1,365,910 tons, of which the Magna plant handled 76 per cent and the Arthur plant 24 per cent. The average grade of the ore was 1.418 per cent copper. The grade is somewhat lower than usual owing to the fact that in steam shovel operations it has been impossible to eliminate all the capping, and some of this has been mixed with the ore. The average cost of copper per lb. during the quarter was 7.85 cents. This item also shows an increase which is ascribed to the lower grade of ore treated and increased cost of operation during the winter months.

The entire Magna plant was operated throughout the quarter, and the seventh remodeled section of the Arthur plant has been put in commission.

Goldfield Consolidated Mines Company.

The annual report of this company for the fiscal year ended October 31, 1911, contains interesting data regarding the remarkable mine and reduction plant at Goldfield, Nevada. During the year 330,549 tons of ore were mined, 330,062 of which were milled and the balance shipped. The net realization amounted to $7,526,846.04. The following tabulations give details of production and expenses.

PRODUCTION.

		Total Value	Per Ton
330,062 tons milled		$10,586,936.42	$32.08
487 tons shipped		173,262.18	355.77
330,549 total production		$10,760,198.60	$32.55
Loss in tailing		597,071.14	1.81
Values realized		$10,163,127.46	$30.74
EXPENSES.			
Mining and development	$1,109,457.95		$3.35
Transportation	30,628.70		.09
Milling	626,293.90		1.89
Concentrate treatment	124,315.70		.38
Marketing concentrate residue and by-products	244,583.94		.74
Marketing bullion	49,141.23		.15
Marketing ore shipped	17,103.92		.05
General expenses	181,334.80		.55
Bullion tax	126,319.26		.38
Income tax	27,001.19		.08
Total operating expenses	$2,536,180.59		$7.66
Miscellaneous earnings	37,730.30		.11
Net operating expenses		$2,498,450.29	$7.55
Realization from operations		$7,664,677.17	$23.19
Construction		137,831.13	.42
Net realization		$7,526,846.04	$22.77

The net realization from operations amounts to 69.95 per cent of the value of the total production. During the year the mill was in operation 96.37 per cent of the total time, and averaged 9.39 tons per stamp actual running time.

CORRESPONDENCE.

The Position of the Induction Furnace.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—I have read with interest a letter in your January issue from Mr. C. H. vom Baur concerning "The Position of the Induction Furnace," in which he takes exception to a statement which appeared in my recent article on "The Grönwall Steel Refining Furnace." This statement that "it would appear that the induction furnaces are on the decline," was intended to refer only to simple induction furnaces and not to those operating with a combination of induction and resistance heating, such as the Röchling-Rodenhauser furnace mentioned in Mr. vom Baur's letter.

The inability of the simple induction furnace to work with dephosphorizing and desulphurizing slags is admitted by the late Dr. Kjellin in his paper before the American Electrochemical Society, May 6, 1909, and further by Mr. Joh. Härden (who was closely associated with Dr. Kjellin in his work) in his recent paper on the Paragon electric furnace read before the Faraday Society, Oct. 2, 1911.

As the present demand is for furnaces which will melt cheap impure raw materials and refine them to produce high-quality steels, I think the future will show greater developments for arc furnaces than for those of the simple induction type.

Toronto, Ontario. T. D. ROBERTSON.

Discussion at San Francisco on Alaska Coal Mines

To the Editor Metallurgical and Chemical Engineering:

SIR:—On page 23 of the January number of METALLURGICAL AND CHEMICAL ENGINEERING, as a part of a so-called "abstract" of the Discussion, at the San Francisco meeting of the American Institute of Mining Engineers, of the proposed operation of a coal mine in Alaska by the Federal Government, occurs, in a summary of my remarks, the following statement:

"One sentence of Dr. Raymond: "You cannot cure national evils," was so brutally pessimistic and so contrary to facts that his further tirade fell on deaf ears."

This quotation may possibly be an accurate *verbatim* report; for I cannot now recall the exact words which I may have used in *extempore* remarks which were to be subsequently replaced by more careful statements, published in the *Transactions* of the Institute. But I am perfectly sure—and my recollection in this particular is confirmed by others who were present, and whom I have consulted—that I did not use the words thus quoted, except as a part of a sentence, the whole of which conveyed a much less offensive meaning.

I was defending the familiar proposition that administration of the local affairs and interests of citizens should be intrusted, as far as possible, to local governments, because these are more accessible to complaints, and more sensitive to the demands of the public affected, than a distant central government could possibly be. In the course of this argument I said that maladministration in a given State could be remedied by such exposure and agitation as would make the people of that State hotly insist upon reform, but that we could not so easily cure such evils if they occurred in Federal administration, because it was so difficult (I may have said "practically impossible") to make the Congressional representatives of 48 States simultaneously indignant over the wrongs suffered by the citizens of one State only.

This is not pessimism, but simply common sense and notorious experience. I could have illustrated my proposition with recent instances of Federal usurpation and injustice, the victims of which were without redress. But I do not propose to offer either proofs or arguments in reply to the remark of your reporter.

My views on the subject of Mr. Bain's paper will be published by the Institute and it is not likely that I shall be tempted, before such publication, to notice again any misrepresentation of them.

New York City. R. W. RAYMOND.

* * *

To the Editor of Metallurgical and Chemical Engineering:

SIR:—If Dr. Raymond, in publishing his discussion of Mr. Bain's paper, repeats the opinions and sentiments expressed in his *extempore* remarks in question, the reporter of the latter is perfectly content to leave the questions of the correctness of the report and of the justification for the criticisms to those who read the *Transactions* of the Institute.

South Bethlehem, Pa. JOS. W. RICHARDS.

Refractories.

BY FRANCIS A. J. FITZGERALD.

In the course of the past eight years the study of various forms of electric furnaces in the FitzGerald and Bennie Laboratories has involved considerable research work on the problem of obtaining suitable refractory materials. Attempts have frequently been made to interest some of those engaged in the manufacture of the ordinary well-known refractories in this problem, but the results have been very discouraging. This, however, is not surprising since in order to meet the special requirements of electric furnaces, and indeed many fuel-heated furnaces, it is necessary to go to the electric furnace for the refractory materials.

Last year the matter of making a study of refractories with the object of solving all sorts of problems of this nature and manufacturing the special materials necessary, was discussed very thoroughly with the Norton company. This company had already devoted a great deal of time to research work on "alundum" as a refractory material, and for some time had been engaged in the manufacture of silicon carbide which it sold as an abrasive under the name of "Crystolon." The pioneer work of Mr. Benjamin Talbot, as well as subsequent experiments by many others, had shown the value of silicon carbide as a refractory, so that the Norton company already had in its factories unusually excellent opportunities for the manufacture of two useful materials for this class of work. It thus seemed that a combination of the facilities of the FitzGerald and Bennie Laboratories and the various manufacturing plants of the Norton company would be a useful one for attacking the refractories problem. It was finally decided to carry on research work on the manufacture and use of certain refractory materials so as to be in a position to solve the problems which would be presented, and as a certain amount of progress has been made recently a brief account of the work done will be of interest.

Some of the refractory materials on which investigations have been carried on are as follows:

Alundum.
Crystolon (Silicon carbide).
  Crystalline.
  Amorphous.
Magnesia.
Lime

An account of some of the work done on the above materials follows:

Alundum.

It is not necessary to deal in much detail with this material as an excellent account of the work done by the Norton company was presented in a paper by Mr. L. E. Saunders at the nineteenth general meeting of the American Electrochemical Society[1]. It may be noted that alundum is made by purifying and fusing bauxite in an electric furnace. There are two forms, white and a brown material, the former being the purer. The white alundum has a melting point between 2050 deg. C. and 2100 deg. C., while that of the brown product is not more than 50 deg. lower. The linear coefficient of expansion of the white variety is 78×10^{-7} and of the brown 85×10^{-7}. The heat conductivity is much higher than fire brick. The specific gravity is 3.9 to 4.0.

Most of the work to which alundum has been applied so far is in laboratory apparatus, such as crucibles, muffles, combustion boats, cores for laboratory electric furnaces, combustion tubes, etc. Recently much experimental work has been done at the Norton Company's Research Laboratories on the production of pyrometer tubes, that is tubes for protecting the thermo-couples of pyrometers, and it is believed that an excellent tube can now be manufactured. The tubes made of alundum are more refractory than any of the porcelain or fire clay tubes, and, what is very important, have a distinctly higher heat conductivity. In using pyrometers in many kinds of work the changes of temperature are so slow that the heat conductivity of the pyrometer tube is not a serious consideration, but in other work it is of great importance to eliminate this lag in the temperature indications. How serious the heat lag may be will be shown below in discussing some of the work done with silicon carbide.

In Mr. Saunders's paper reference was made to experiments with alundum bricks. That these bricks are highly refractory there can be no doubt whatever. In the Norton company's experimental department a furnace roof was made of alundum brick and the remarkably high temperature these would stand without giving way demonstrated the point beyond doubt. The temperatures used in the experiments were so high that a roof built of silica brick was destroyed in five or six hours, while 40 or 50 heats at similar temperatures with the alundum roof failed to make it useless. On the other hand experiments with roofs built of alundum brick in a steel furnace did not give results which came up to expectations, although the temperatures were by no means so high as in the furnace referred to above.

The problem of constructing a good roof for a steel furnace of the surface arc type has been the subject of careful study at the FitzGerald and Bennie Laboratories in conjunction with the Research Department of the Norton company. Alundum brick made up in various ways have been tested in a small single-phase, two-arc steel furnace. These experiments have clearly shown that while alundum bricks will successfully resist the temperature of the electric steel furnace roof there are difficulties in the way of using them for that specific purpose. The bricks when used in the steel furnace roof showed a tendency to break off in layers and careful analyses soon showed that this was due to the action of the lime vapors rising from the intensely heated basic slag used in the furnace. The action in fact was altogether analogous to that pointed out by Hansen in his discussion of Saunders's paper, showing that in the steel furnace the stalactites which formed on the roof contained a relatively large percentage of lime and magnesia. The lime and magnesia vapors at high temperatures react with the alundum brick and thus shorten the life of the roof. The final conclusion reached, as a result of the study, was that while alundum brick could be made which would certainly outlast silica brick when used as a furnace roof, the increased life was not sufficiently great to warrant the extra cost of the special refractory in this particular case.

Crystolon.

This is the name given to the silicon carbide manufactured by the Norton company. It is obtained in two forms, crystalline and amorphous. The earliest work which has been published on silicon carbide as a refractory material was by Benjamin Talbot, who obtained a broad patent on the use of the material in that way.[2] Talbot says[3]:

"I have found that carbid of silicon (which in the crystalline form is commonly called 'carborundum') if formed into bricks or compressed into a lining for metallurgical structure, is very refractory under high temperatures and resists the fluxing action either of basic or acid slags used in steel, iron or other metal manufacture . . . Either the crystalline or amorphous carbid may be used, preference being given to the amorphous carbid because that has hitherto been deemed a waste product and is consequently inexpensive."

Talbot in general proposes to use a binding agent such as tar, but does not limit himself "to any particular method of forming the lining," but claims, broadly, the use of carbid of silicon however applied as a refractory lining for metallurgical structures.

[1] Transactions of the American Electrochemical Society, Vol. XIX (1911), p. 333.
This journal, Vol. IX, p. 257; May, 1911.

[2] U. S. Patent No. 628,288.

"In making repairs to the lining the carbid may be used dry or moistened with water or mixed with any appropriate binding agent, according to the nature of the repairs."

The amorphous carbide referred to by Talbot is, what may be called, the first stage in the formation of crystalline silicon carbide. When a mixture of sand and carbon is slowly heated to the point where a reaction occurs there is formed a greenish colored mass showing no crystalline structure, but consisting in large part of amorphous silicon carbide. As the process of conversion is gradual the product as taken from commercial furnaces is never chemically pure. At the outer parts of the charge, that is the parts furthest from the source of heat, there will be unconverted mixture mixed with the green-colored amorphous substance and near to the source of heat the amorphous substance will gradually grade into crystalline silicon carbide. Moreover, in the process of conversion it is possible that certain silico-carbides of the formula $Si_xC_xO_y$ are formed;[3] these corresponding to Acheson's "silioxicon." If, however, a fair sample of the amorphous material is obtained and purified by acid treatment, etc., an amorphous substance having the analysis corresponding to silicon carbide is obtained. A specimen prepared in this way gave the following analysis:

Silicon	69.41
Carbon	29.18
Iron and aluminium	0.59
	99.18

For a refractory it is, of course, unnecessary to have this specially purified material, the only necessary precaution being the avoidance of an excess of silica which might be present in unconverted mixture used in the furnace charge. Talbot calls attention to this point in his patent.

Since Talbot's original work on this material it has found a considerable market for use in various kinds of furnaces, etc., in the form of "carborundum fire sand." As a lining in various forms of resistance furnaces it gives most satisfactory results. For example, when it is desired to heat crucibles by embedding them in a resistor composed of granular carbon, trouble is often experienced at high temperatures by the melting of the fire brick forming the sides of the trough containing the resistor. If now the brick trough is made a little wider and then lined with amorphous silicon carbide to a thickness of 1 in. no trouble at all is experienced.

In many fuel-fired furnaces this refractory also gives excellent service in the form of bricks or linings, or even as a wash applied to the fire brick walls. It can be usefully applied in making repairs to furnaces and as a cement for setting up fire brick.

When this amorphous substance is heated to excessively high temperatures it becomes somewhat plastic and eventually in a reducing atmosphere is converted into the crystalline variety of silicon carbide. The fact that it becomes somewhat plastic at very high temperatures makes it possible, in special cases, to use it as a lining, etc., without any binding agent whatsoever, as indicated in the Talbot patent.

Amorphous silicon carbide is not a good heat insulator. Its heat conductivity while not very high is, nevertheless, distinctly greater than that of ordinary fire brick structures.

Formed at a higher temperature than the amorphous form, crystalline silicon carbide, or "crystolon," is a still better refractory material for many purposes. Under ordinary conditions it is infusible, but when heated to a sufficiently high temperature in a reducing or neutral atmosphere it is decomposed, silicon vaporizing and the carbon remaining behind as graphite. If heated to a sufficiently high temperature in a strongly oxidizing atmosphere, slow oxidation occurs, the carbon burning to carbon-monoxide and the silicon forming silica. The latter appears to form a protective coating which prevents further oxidation, or at least prevents it from proceeding rapidly.

Experiments have been carried out in the Norton company's Research Department on various articles made of crystolon grains or powders mixed with suitable binding agents, and so far the results obtained are found to be highly satisfactory; but more time and work have been expended on the study of crystolon made into various forms without the use of any binding agent. These articles are made according to a method patented by FitzGerald.[4] Silicon carbide in the form of grains or powder is mixed with some temporary binder, such as a solution of glue or dextrine, the mixture molded in the desired form and then heated in an electric furnace to the temperature at which silicon carbide is formed. This causes a crystallization or recrystallization of the silicon carbide and a strong very refractory article is obtained. These are known as "pure crystolon" bricks or tubes or whatever form these may be in order to distinguish them from bonded crystolon articles.

Most of the experiments which have been made with pure crystolon have been with bricks made of that material. As was already pointed out the bricks are infusible under ordinary conditions. The coefficient of expansion by heat is small, much less than that of silica for example. The heat conductivity is higher than that of alundum even. This is well shown in the figure where curves are given which illustrate the relative capacity for transmitting heat of silica, alundum and pure crystolon bricks. The method of obtaining the data from which the curves were plotted was to build a small furnace of the bricks under test and to place therein a resistor through which a measured current could be passed. Through a small hole in the wall of the furnace a thermo-couple was passed so that the temperature of the interior could be measured. The furnace was then heated with a definite current and when the temperature was constant both it and the watts were noted. This gave the amount of energy passing through the walls of the furnace for that particular internal temperature. In the curves the abscissae are temperatures in the furnace and the ordinates are watts, giving in that unit the rate at which heat is flowing through the walls. Curve 1 is for the pure crystolon brick, curve 2 the alundum brick and curve 3 a silica brick. It is very clear that the capacity of the crystolon brick for transmitting heat is very great compared to other

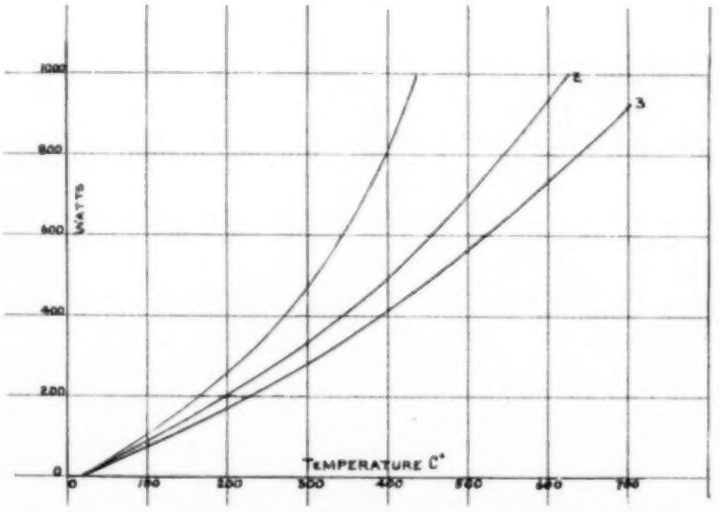

FIG. 1.—COMPARATIVE HEAT CONDUCTIVITIES OF (1) CRYSTOLON, (2) ALUNDUM, (3) SILICA.

[3] *Electrochemical Industry*, Vol. II (1904), p. 439 *et seq.*

[4] U. S. Patents Nos. 650,234 and 650,235

refractories. Thus when the temperature in the furnace is 435° the crystolon brick transmits 79 per cent more heat than the alundum brick and 117 per cent more than the quartz. The crystolon brick is, therefore, a very efficient refractory where it is desired to transmit heat through it; but on the other hand where good heat insulation is necessary it is important to back up the crystolon with a suitable heat insulator.

There have not been any thoroughly satisfactory quantitative experiments made on a large scale to study the efficiency of crystolon as a transmitter of heat, but two tests may be mentioned which give some indications on this point. A furnace having a capacity of 75 to 100 kw was constructed, in which the resistor heated the reaction chamber by radiation. Between the resistor and the reaction chamber was a septum built of crystolon tile 1 in. thick. Temperature determinations made with a thermo-couple on both sides of the septum showed a very small difference in temperature when the furnace was at high temperatures. In this relation, the form of curve 1 in the figure should be noted, as this indicates that the heat conductivity increases in marked degree with increased temperature.

The other experiment was a comparative one made with two pyrometer tubes, one made of a refractory clay and the other of graphite and crystolon. The dimensions of the tubes were identical, but the special tube was built up of two tubes the inner one being of graphite and the outer one of crystolon. The experiment was made by introducing the cold tubes into the furnace, the temperature of which was kept constant, and then noting the time taken for the pyrometer to come to a constant reading. In the case of the clay tube the time was 38 minutes, while the graphite-crystolon tube reached a constant temperature in 15 minutes. It was, moreover, observed that the clay tube gave a pyrometer reading 10 deg. lower than the other.

The most important study that has been made of the use of pure crystolon is in the building of the roofs of electric furnaces. The experiments have been made both in the Fitz-Gerald and Bennie Laboratories, where the crystolon brick were manufactured, and also in other works. The laboratory furnace was a small one having a capacity of about 500 to 600 lb. of steel, but it was possible to use a current as great as 150 kw. Moreover, in constructing the furnace the hearth was built very high so that in some cases the surface of the bath was only 8 in. distant from the roof of the furnace. This, of course, made the test a very severe one.

Certain characteristics of pure crystolon makes it necessary to construct the roof of the furnace in a special way. In the first place the high heat conductivity of pure crystolon would cause serious losses from a roof built in the ordinary way. This was clearly demonstrated by building a roof of pure crystolon brick, 9 x 4½ x 2½ in., with the brick placed so that their longest dimensions were vertical. When the furnace was worked on steel in this way it was impossible to heat it properly on account of the rapid flow of heat through the roof. After that the roof was built in two courses, the lower one consisting of crystolon brick laid so as to form a roof 4½ in. thick, and the second course consisting of fire brick also 4½ in. thick. This was found to be a perfectly satisfactory arrangement. It has also the advantage that the number of crystolon brick required to form a roof is reduced one-half in this manner.

The other characteristic of crystolon which makes a special construction necessary is its electrical conductivity at high temperatures. This phenomenon can be well illustrated by a simple experiment. If a rod of pure crystolon is wound with copper wire at both ends so as to make a good contact and connected to a source of electrical current, very little will pass, but if the rod is heated with a blast lamp to a high temperature the current will pass freely. Crystolon refractory articles made up with a small amount of a clay bond do not become conductors even at high temperatures, although the resistivity is considerably diminished. The following determinations of the resistivity of bonded crystolon have been made:

Temperature C°.	Resistivity (cm. cube)
20	318 x 105
656	63 x 105
809	32 x 105
940	16 x 105
1040	4 x 105

It will be observed that although there is a considerable decrease in resistivity with rise in temperature, yet for most practical purposes the crystolon remains an insulator.

If now the pure crystolon brick were used to build an electric furnace roof through which two electrodes at different potentials passed there would be serious current leakage when the roof becomes hot if the electrodes touched it. It is, therefore, necessary to have a clearance between the electrodes and the brick by making the openings in the crystolon roof somewhat larger than is customary with the ordinary roof. On the other hand the heat insulating brick used on top of the crystolon roof form openings that are of the normal size. This method of constructing the roof is found to be perfectly satisfactory.

In the tests made at the FitzGerald and Bennie Laboratories it was found that when using ordinary brick and overloading the furnace there was no difficulty in completely destroying the roof in a single heat, but this was impossible in the case of the crystolon roof. When the crystolon roof is used it is possible so to overload the furnace that the magnesite lining is destroyed and yet the roof remains intact.

Experiments have also been made on 2-electrode steel furnaces in which the capacity was 2.5 tons of steel, and these in every respect confirm the results already mentioned.

It is certain that, provided a sufficiently high temperature is reached, there is a reaction between lime and silicon carbide, but so far as the tests made with the crystolon roofs go there is no appreciable reaction between the lime vapors from the slag and the brick.

The pure crystolon brick has been tested also in other ways. Thus, in one furnace in which a certain material was heated to excessively high temperature, it was found that a good grade of fire brick used outside the crystolon brick for heat insulating purposes could be melted down completely by the heat transmitted by the crystolon brick without in any way damaging the latter.

Other examples could be given, but enough has been said to show that this refractory material promises to have important uses, making it possible to do things which were impossible hitherto. For example, in the electric steel furnace not only may the space between the roof and the bath be made much smaller but the temperature may be forced to a higher degree without any risk of damage.

The crystolon brick is, of course, a somewhat expensive refractory and consequently much of the investigating work done on it has been in the direction of improving methods of manufacture so as to reduce the cost. So far as molding goes the problem is comparatively simple, but the development of a satisfactory furnace for "baking" the brick proved more difficult. However, a satisfactory furnace has been designed and worked until it appears to give excellent results.

Electrically Calcined Magnesia.

For many purposes, as is well known, magnesia is a most excellent refractory, but it has also serious faults. The most serious of these is an objectionable way of cracking when heated to a high temperature. This is, no doubt, in large part due to the remarkable way in which magnesia articles shrink when heated to high temperatures, corresponding to the increase in the absolute density. Moissan has described an experiment on this subject which is worth quoting here:

"The first specimen (of magnesia) had been heated for 6 hours in a furnace heated with an air blast and retort carbon as fuel. The second specimen consisted of magnesia plates, partly crystalline, which had been submitted for two hours to the action of the electric arc. The third consisted of a mass of magnesia weighing 50 grams and fused into a single piece in the electric crucible furnace.

"We obtained the following:

	Density at 20°
"1. MgO (furnace with blower)	3.577
"2. MgO (furnace plates)	3.587
"3. MgO (fused mass)	3.654"[5]

Moreover, Ditte has shown that magnesia gives the following densities:

	Density at O°
At 350°	3.1932
Dull red	3.2482
White heat	3.5699[6]

Under the circumstances it is not surprising that articles such as brick and crucibles are apt to crack when heated to high temperatures.

It is obvious that the tendency to cracking exhibited by the ordinary calcined magnesite is less likely to be found in a material which has been calcined at the very much higher temperatures attainable by electrical means. An experiment which brought this out very clearly was as follows: A crucible was made of highly calcined magnesia made from Grecian magnesite. The crucible was embedded in a granular carbon resistor and the current regulated so as to heat up the crucible very gradually. The crucible was finally raised to an intense white heat, probably about 1500° C., and was then allowed to cool down gradually in the furnace. It was so badly cracked that it was absolutely useless. Another crucible was made in exactly the same way except that electrically calcined magnesia was used. This crucible was heated fairly rapidly to a temperature at which Seger cone N° 35 (supposed to bend at 1830° C.) was completely fused. The crucible was then allowed to cool down and was examined. A few very small cracks had occurred where a hot spot unfortunately developed during the process of heating, but otherwise the crucible was in perfect condition. The heat had been so intense that the outside of the crucible was slightly fused.

Several tons of magnesia have been electrically calcined and tested in a great variety of furnaces, and in every case excellent results have been obtained.

There is one other important point to note in regard to this highly calcined material. The ordinary calcined magnesia absorbs carbon dioxide gas forming magnesium carbonate more or less rapidly, but this does not occur at all, or with extraordinary slowness, in the case of the electrically calcined product. An experiment demonstrated this very clearly. Two porcelain boats were filled with electrically calcined magnesia and a highly calcined magnesia of the ordinary kind, respectively. In both cases the samples were finely ground so that the particles were of approximately the same size. The boats were placed in a tube through which a rapid stream of moist carbon dioxide gas was passed for 44 hours. At the end of that time the samples were weighed with the following results:

Sample.	Per cent increase in weight.
Ordinary calcined	31.30
Electrically calcined	0.00

In the preparation of this magnesia it is possible to use an arc furnace, but the method is not altogether satisfactory on account of its cost. It is unnecessary, and in fact disadvantageous in many ways, to fuse the magnesia. Accordingly a resistance furnace has been worked out which will calcine the magnesia to the desired degree.

[5] Moissan "Le Four Electrique," Paris 1897, p. 35.
[6] Comptes rendus t. LXIII, p. 111 and p. 191.

Fused Lime.

Some hundreds of pounds of lime have been fused in the electric furnace and an interesting product obtained. Little work has been done on the use of this material as a refractory, but certain of its properties indicate that it is worth while studying it. It appears that in this form it is a much better conducor of heat than ordinary quick lime. Blocks cut from the fused product stand heating followed by sudden cooling very well. One such block was heated to redness with a blast lamp and while still under heat cold water was poured on the block without producing any change. Fused lime resists exposure to moist air remarkably well, for while it will eventually slack this is a process of days instead of hours. Even when boiled with water, hydration proceeds very slowly.

Conclusion.

Other refractory materials are under investigation and some will undoubtedly give interesting results. The study of the subject indicates very clearly that every refractory problem requires individual study for its solution, and that for at least some of these problems suitable refractory materials have already been discovered.

FitzGerald and Bennie Laboratories,
Niagara Falls, New York.

Concentration by Flotation of Dry Ore Pulp.

By H. C. Parmelee.

The systems of concentration by flotation that have been exploited in the past decade exhibit a variety of applications of one or two underlying principles. The use of oil and acid mixtures, with or without the application of a vacuum, is common to most of them; and with the exception of the method to be described herewith, all of them treat pulps which have been submerged in water. In some of them the floating concentrate is formed as a thin film, approximately one particle deep, while in others it gathers as a thick scum on the surface of the liquid in the treatment tank.

The method of flotation here described is the invention of Henry E. Wood, of Denver, Colo. It falls within the class of film flotation processes, but is distinguished from these and all others by the fact that neither oil nor acid is used, and that the ore pulp is floated in the dry state just as it comes from the crushing machine. The basis of the separation lies in the difference in the effect of gangue and mineral particles on the surface tension of water.

Before proceeding to a dscription of the machine and its operation some consideration of its principles may not be amiss. The working of the process involves the physical properties of cohesion and adhesion and their attendant phenomena of surface tension and capillarity. It is well known that a liquid behaves as though its surficial film is in a state of tension tending to contract. This is usually ascribed to the molecular attraction of the liquid particles. Under normal conditions water exhibits this phenomenon of surface tension in a very high degree, more highly, perhaps, than any other liquid except mercury. The state of tension on the surface, however, is modified or affected by the presence of a solid particle in contact with the surface of the liquid. The surface of water, for example, near the sides of its containing vessel, or surrounding any object placed on its surface, is not truly horizontal. If the solid is wetted by the water, the surface of the liquid is slightly concave in that vicinity; if the water does not wet the solid, the liquid surface is slightly convex along the line of contact. The concavity and convexity are capillary phenomena and depend respectively on the relative molecular attraction of the liquid particles for each other (cohesion), and the attraction of the liquid particles for the solid (adhesion).

In case the adhesion between the solid and liquid is greater than the cohesion of the liquid particles, we find capillary elevation of the liquid surface and a wetting of the solid. If the cohesive force is the stronger, we find depression of the liquid surface and the solid is not wetted. This action is confined to thin surficial films or layers of the solid and liquid. The law[1] is thus stated: The liquid will be elevated in a concave curve or depressed in a convex curve by the side of the solid according as the attraction of the liquid molecules for each other is less or greater than twice the attraction between liquid and solid.

The temperature of the liquid exerts an effect on the action of the adhesive and cohesive forces, increasing the capillary action of the liquid surface as the temperature rises. In general, when solids are specifically heavier than the liquid, those which become wetted will sink and those not wetted will float.

Minerals Which Float.

An investigation of the principles underlying flotation processes was published[2] in 1911 by Kenneth A. Mickle. Among other points he studied the behavior of minerals and metals toward cold and hot water. He found that when minerals are carefully deposited on the surface of water many of them float, while others sink immediately; also that considerable weights could be supported by a thin film of floating mineral. He gives the following list of minerals which floated on the surface of cold water: Galena, blende, sulphur, azurite, malachite, rhodonite, garnet, calcite, mica, telluride of gold, pyrite, pyrrhotite, wolfram, cassiterite and serpentine. The list is open to question from a commercial standpoint, however, for apparently the tests were made on a small surface of water, e.g., on a cylindrical graduate or beaker, where the surface tension did not have free and unobstructed play. If the tests

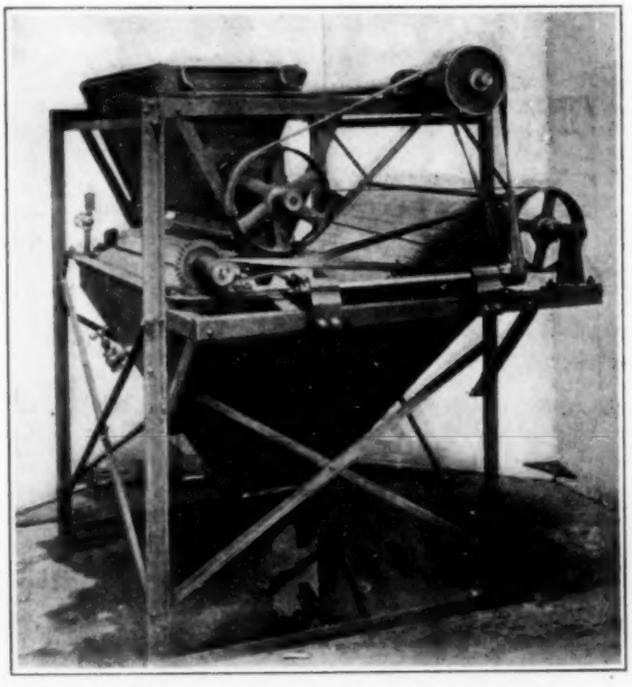

FIG. 1.—WOOD FLOTATION MACHINE.

had been made on a large surface of water, say 10 sq. ft., there is little doubt that some of the minerals mentioned would have sunk. This will be indicated later.

On hot water the behavior of the minerals was different, most of them becoming wetted and sinking after floating for a few seconds only. This apparently was due to the decreased surface tension of the water.

The Wood Machine.

The Wood machine for flotation of dry-crushed ore is illustrated in two views in Figs. 1 and 2. Fig. 1 shows especially the feed, and Fig. 2 the discharge end. The machine consists of an angle-iron frame supporting a feed hopper, shaking feed plate, partly submerged feed roller, belt conveyor for removing concentrate and tank of water on the

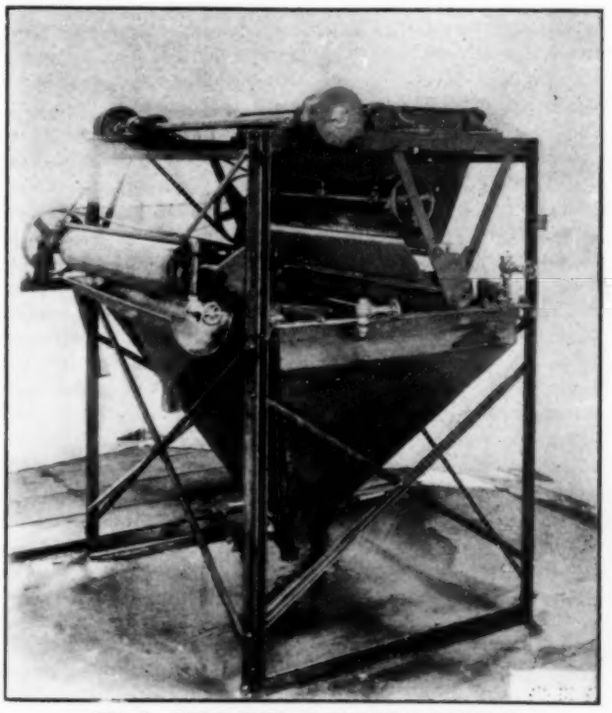

FIG. 2.—OTHER VIEW OF FLOTATION MACHINE.

surface of which flotation occurs. The floor space occupied is about 4 ft. by 6 ft. and the height 6 ft.

Referring to Fig. 1: The dry ore, suitably crushed, is fed from the hopper onto a shaking tray, which in turn delivers a thin stream of ore onto a partly submerged revolving roller, shown at the left of the tank. The roller is covered with some form of corrugated rubber, the size of the corrugations depending on the mesh of ore under treatment, being larger for coarse pulp. As the roller revolves (clockwise as seen in the figure) it elevates a certain quantity of water. The ore pulp falling onto the roller in a thin stream is thus immediately exposed to the wetting action of water, and is then delivered onto the surface of the water in the tank. The gangue particles become wetted by capillary action and sink, while the mineral concentrate tends to float and soon covers the entire surface of the water. In order to facilitate the movement of the film of concentrate across the tank to the belt conveyer gentle streams of water from submerged perforated pipes are directed in a suitable manner against the under side of the film.

On reaching the discharge side of the tank the concentrate is removed by a belt conveyer, the tail pulley of which is below the water surface in the tank. The point of discharge is better shown in Fig. 2, where the head pulley appears above and outside of the tank. Water from a perforated pipe washes the concentrate off the belt into a hopper. The flotation tailing sinks in the large tank, whence it can be removed, if desired, for subsequent treatment on concentrating tables or discharged as clean tailing.

In the course of Mr. Wood's experiments many interesting facts have been discovered regarding flotation of minerals on water. He finds that the sulphides generally exhibit a more or less marked tendency to float, as also do the tellu-

rides, graphite and metallic gold and silver. These substances are not easily wetted, and consequently cannot get beneath the surface of the water. The sulphides of copper, lead, silver and molybdenum, for example, strongly tend to float, while those of iron and zinc show this property in a less marked degree. Thus preliminary separation of some minerals can be made by the flotation process, as will be shown later. In the case of a mineral which shows a strong tendency to float, e.g., tetrahedrite it is possible by preliminary flotation to make a recovery of mineral which in the ordinary processes of concentration would be lost. In extremely favorable instances, as in the case of molybdenite and graphite, it is possible to produce concentrates that are not recoverable by ordinary methods of concentration.

Oxidized minerals and gangue substances generally tend to sink when brought onto the surface of water, becoming wetted by capillary elevation of the liquid surface and ultimately being completely immersed. In this respect the list of floating minerals given by Mr. Mickle is apparently at fault, for he includes azurite and malachite, neither of which is commercially amenable to direct flotation.

The mode of preparation also affects the results obtainable by this system of flotation. The degree of fineness to which an ore should be crushed in order to secure best results can be determined only by experiment; but in regard to the manner of preparation there seems to be little doubt that the ore should be *crushed* and not *ground*. The ball mill and rolls yield a more suitable product than do the tube mill and disc grinder. In the former machines we get a fracture due to a crushing blow, while in the latter we get reduction by abrasion. The crushers produce tabular particles whose surfaces are more cleanly fractured, while the grinders produce surfaces that are irregular and less well defined. The grinders also produce an excessive amount of fine mineral, thereby increasing possibility of loss. Inasmuch as this form of flotation is vitally concerned with the relations of surfaces, it is not difficult to see wherein one method of preparation might be superior to the other.

Another point which is indicated in some experiments, although not as yet fully determined, is that long exposure of crushed ore to the air is detrimental to flotation. In other words, the best results are obtained from freshly crushed ore. Perhaps incipient oxidation of the sulphide surface is sufficient to destroy its effect on surface tension; but whatever the cause may be, the results are as stated.

As with other forms of concentrating apparatus, the capacity of this machine varies with the ratio of concentration. If the ore contains much mineral which will float the capacity is naturally lower than in the opposite case. The surface of the water must not be crowded with concentrate, as otherwise the mineral particles may become wet and sink. Over a wide range of ores, however, the capacity has varied from 1000 to 2000 pounds per hour.

Results of Tests.

The results from a series of tests made specially for the writer are given below. The ore contained chalcopyrite and pyrite, silver being associated with the former. For all tests except No. 8 the ore was crushed in a Krupp ball mill. For No. 8 the ore was ground in a disc pulverizer. The effect of ball mill crushing in producing tabular mineral fragments was specially noticeable. In reconcentrating the flotation tailing on a Wilfley table thin tabular pieces of pyrite were observed to float like mica and pass into the tailing. The gangue associated with the copper and iron minerals in this ore was very heavy, apparently containing a manganese mineral. As a result it appeared to a large extent in the Wilfley concentrate, making a low grade silicious product. A screen analysis of the flotation concentrate from the test on 16-mesh pulp showed that 24.7 per cent of the weight of that product was between 16 and 40-mesh in size, thus showing that not only fine but appreciably coarse mineral will float by this process. The fact that the series was discontinued at 16-mesh size must not be construed as indicating the coarsest pulp that can be floated. As a matter of fact, 10-mesh pulps can be treated successfully; but at 16-mesh we practically reach the coarse-grade limit of table concentration.

Test No. 1.—Ore ground to 30-mesh, mixed with water and concentrated without sizing on a Wilfley table only:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 30-mesh	2268	2.22	1.50	....	....
Wilfley conc (9:1)	252	7.90	6.28	39.53	46.50
Wilfley tail	1932	1.40	0.88	53.84	50.14
Not accounted for	84	...	...	6.63	3.36
				100.00	100.00

This shows the inefficiency of ordinary wet dressing on this ore. More than half of the silver and copper appear in the tailing.

Test No. 2.—Ore ground to 40-mesh and treated by flotation only:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 40-mesh	2268	1.50	....
Flotation conc. (23.14:1)	98	20.60	59.34

The ratio of concentration, grade of concentrate and percentage of recovery are higher than on a Wilfley table.

Test No. 3.—Ore ground to 30-mesh and concentrated by flotation. Reconcentration of flotation tailing on a Wilfley table without preliminary sizing:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 30-mesh	2268	2.80	1.51	....	....
Flotation conc. (21:1)	108	23.10	20.42	39.28	64.82
Wilfley conc. (24.65:1)	92	12.30	2.02	17.67	5.46
Total conc. (11.34:1)	200	18.45	11.95	56.95	70.28

This shows higher copper recovery at 30-mesh than in test No. 2 at 40-mesh. Additional small recovery is also made by the Wilfley table.

Test No. 4.—Noting the higher copper recovery by flotation at 30-mesh as compared with 40-mesh, this test was made on 20-mesh pulp to see whether additional improvement would result from coarser crushing. The flotation tailing was reconcentrated on a Wilfley as before:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 20-mesh	1900	5.13	....
Flotation conc. (16.96:1)	112	30.28	34.80
Wilfley conc. (7.27:1)	261	17.33	46.47
Tailings	1527	1.20	18.79
Total conc. (5.09:1)	373	21.22	81.27

This shows further improvement in the grade of flotation and table concentrates, as well as higher percentage of recovery.

Test No. 5.—This was made on pulp passing 16-mesh to ascertain the effect of still coarser crushing. The flotation tailing was reconcentrated as before without sizing:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	1500	6.01	....
Flotation conc. (13.88:1)	108	30.60	36.68
Wilfley conc. (5.43:1)	276	17.42	53.38
Tailings	1116	0.82	10.79
Total conc. (3.9:1)	384	21.13	90.06

Here we find about the same grade of concentrates as in the preceding test, but with about 9 per cent greater recovery.

Test No. 6.—This was made with 16-mesh pulp in a similar manner to test No. 5, except that the flotation tailing was sized for concentration on the Wilfley table:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 16-mesh	2268	7.80	7.30	....	....
Flotation conc. (5.46:1)	415	23.70	26.35	55.65	66.10
Wilfley conc. (4.5:1)	503	8.40	8.95	23.93	27.24
Tailings	1247	...	0.62	....	4.69
Total conc. (2.47:1)	918	15.31	16.81	79.58	93.34

The total recovery is slightly increased over that of test No. 5.

[1]Olmsted's College Philosophy.
[2]Australian Mining Standard, April 5, 12, 20, 27, 1911.

Test No. 7.—Similar in all conditions to test No. 6. The recovery of silver is appreciably higher. The flotation concentrate contained but 3.88 per cent silica:

	Weight grams	Assay Ag. oz.	Assay Cu. %	Distribution Ag. %	Distribution Cu. %
Ore taken, 16-mesh	4536	5.20	4.50	....	....
Flotation conc. (17.23:1)	627	27.90	24.25	74.16	74.54
Wilfley conc. (6.48:1)	699	7.60	4.28	22.52	14.67
Tailings	2965	...	0.35	....	5.08
Not accounted for	....	...	...	....	5.71
Total conc. (3.42:1)	1326	17.20	13.72	96.68	89.21

Test No. 8.—This test was made to show the effect of grinding as compared with crushing. The ore was ground to 16-mesh in a disc pulverizer and then concentrated by flotation. The results are to be compared with those of the other tests on 16-mesh pulp. The difference in recovery is not as great at 16-mesh as it probably would be with finer pulps:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	2268	6.25	....
Flotation conc. (5.54:1)	409	23.02	66.49

The recovery is lower than that by flotation in test No. 7 and about the same as that in test No. 6.

Test No. 9.—As a matter of interest in connection with these tests one was made using a film of oil on the surface of the water. The ore was floated dry in the usual manner, the oil merely acting as a transporting medium:

	Weight grams	Assay cu. %	Distribution cu. %
Ore taken, 16-mesh	1000	4.50	....
Flotation conc. (6.53:1)	153	23.05	78.45
Wilfley conc. (10.2:1)	98	4.40	9.58
Total conc. (4:1)	251	15.76	88.03

The oil film apparently has no beneficial effect, as the results are not quite as good as those obtained in tests 5, 6 and 7. The flotation concentrate in this test contained but 4.04 per cent silica.

Test No. 10.—This is given to show the results obtained on a wholly different ore. A high-grade telluride ore was crushed to 40-mesh and treated by flotation with reconcentration of the tailing on a Wilfley table:

	Weight grams	Assay Au. oz.	Assay Ag. oz.	Distribution Au. %	Distribution Ag. %
Ore taken, 40-mesh	500	39.74	10.00	....	....
Flotation conc. (14.7:1)	34	506.20	101.70	86.62	69.04
Wilfley conc. (31.25:1)	16	67.44	27.30	5.43	8.73
Tailings	379	4.68	1.80	8.91	13.62
Total conc. (10:1)	50	365.79	77.80	92.05	77.77

Summary of Copper Tests

Treatment	Preparation	Recovery cu. %	Grade cu. %	Conc. ratio
1 Wilfley alone	Crushed 30-mesh	46.50	6.28	9.00
2 Flotation alone	Crushed 40-mesh	59.34	20.60	23.14
3 Flotation and Wilfley	Crushed 30-mesh	70.28	11.95	11.34
4 Flotation and Wilfley	Crushed 20-mesh	81.27	21.22	5.09
5 Flotation and Wilfley	Crushed 16-mesh	90.06	21.13	3.90
6 Flotation and Wilfley	Crushed 16-mesh	93.94	16.81	2.47
7 Flotation and Wilfley	Crushed 16-mesh	89.21	13.72	3.42
8 Flotation alone	Ground 16-mesh	66.49	23.02	5.54
9 Flotation and Wilfley	Crushed 16-mesh	88.03	15.77	4.00

These tests indicate that with particular ore possibly all the chalcopyrite would float. The fact that the tests do not fully bear out this belief may be accounted for by the accidents of manipulation in small tests.

Electrolytic Determination of Lead.

BY ELMER LIST.

The Guess-Haultain electrolytic assaying outfit is a neat device for the rapid and accurate determination of lead in large and small amounts.

The accuracy of this method ranks with that of the most carefully made determinations by other methods and its great advantage is the small amount of the chemist's time required. In the presence of manganese or antimony it is necessary to have a large excess of free nitric acid in the electrolyte, under which condition neither element interferes. Bismuth, even in the presence of large amounts of free nitric acid, is partly precipitated as bismuth oxide with the lead. Its presence can be recognized by a light blue color given to the peroxide coat. Arsenic and tellurium must be removed before electrolysis, for if present they effectually prevent the deposition of lead.

Unless the anode is sandblasted, only a comparatively small amount of peroxide will adhere, but when properly sandblasted, adherent deposits of 250 mg. and even up to 600 mg. of lead peroxide may be obtained in daily work.

A weight of 0.855 mg. is advised in order to avoid the necessity of calculating each result, the weights of peroxide found being called lead. The theoretical factor is 0.866, but in practice 0.855 is found to be more accurate, probably due to an excess of oxygen in the peroxide.

The sample for analysis is weighted into a tall battery beaker of 200 cc. capacity, is digested with 25 cc. of nitric acid. The lead sulphate formed is readily dissolved after boiling, by the addition to the beaker of 25 cc. of a saturated solution of ammonium nitrate containing 25 per cent of free ammonia. This must be added carefully to prevent boiling over. After the solution is complete, the beaker is filled with warm water. The solution is now ready for electrolysis. A wide range of current strength is permissible, but from 1.5 to 2 amp. is the most satisfactory. This amount of current keeps the solution sufficiently warm during the electrolysis. At the end of two hours the anode is removed, washed in water, then alcohol, ignited and weighed.

When the amount of lead in the sample is small (below 0.5 per cent) not more than 5 cc. nitric acid must be present in the electrolyte to secure a complete deposition of peroxide. When more than 25 per cent of lead is present, 50 cc. of nitric acid is necessary to insure a hard, firm, and closely adherent coat. This deposit will not rub off nor crack if enough nitric acid has been used.

To clean the electrodes, they are placed in boiling nitric acid, in which some copper rods have been dissolved, for a few hours.

The electrodes are stationary and of two forms. The anode is shaped into a blade having an immersion area of about 50 sq. cm. and weighs about 1.5 grams. The blade is sandblasted and corrugated lengthwise. Strips of platinum foil form the cathodes. Three electrodes are used in each cell, one cathode on each side of the anode.

These electrodes are connected with slotted aluminum terminals, in which they are held by contact pressure. The terminals are 5/16-inch rods projecting 2 in. horizontally in front of the wall of the cabinet. Below the projecting rods are arranged suitable moveable shelves for holding the beakers during the electrolysis. At the back the anodes are connected with one pole of the current and the cathodes with the other pole.

This method fills the demand for greater speed and higher accuracy in making daily assays of products containing but a small amount of lead and has supplanted the older analytical processes.

St. Louis Smelting & Refining Company,
St. Francois, Mo.

Calculation of Furnace Charges--III.

BY REGIS CHAUVENET.

The main object of this presentation of a "slag method" is to show how, from given materials, to compute a "required" slag of definite composition. Hence we do not go into lengthy discussions of the physical characteristics of various slags, nor of the conditions which make a certain type of slag desirable.

It is necessary nevertheless to recognize the fact that limitations are sometimes imposed by the composition of available fluxes, or of the ore itself, so a few remarks are appended to show how far it is safe to vary the composition, especially as to the limits between which the silica ratio may be varied, and to some extent the proportions which may be allowed of certain bases, and the ratio of the latter "inter se."

Iron.—Almost invariably the only addition for flux in the iron blast furnace is limestone. This may contain more or less magnesia, which, however, is not detrimental except under conditions as to alumina percentage to be presently specified. The ores of iron are low in slagging bases (other than iron), with the exception of alumina, and this, at many iron works is considered in the light of an acid.

Manganese presents no difficulties, the metal will usually be reduced in some measure, entering the pigmetal, another portion joining the slag of course in an oxidized condition. Discussion of the operations necessary for the manufacture of "spiegel" would lead us too far from the intent of this article.

As for lime and magnesia *in the ore*, they serve simply to reduce weight of the limestone to be added as flux.

We are not yet giving actual slag compositions, but may state that while it has been customary in the past to figure iron slags as pure silicates, it is to-day quite as common to assume a certain ratio as between "acid" and "base" and to take the sum of silica and alumina as the "acid." For details of this procedure in actual computations, see examples to follow.

In ordinary cases, with low alumina, the ratio of a singulo silicate may be safely followed, treating alumina as a base. Owing to limitations imposed by materials, little regard can be had to the proportions of the bases to one another, but one point should be noted, viz.: the danger of slagging off iron oxide. No superintendent of any knowledge or experience would incur such a danger, but a brief discussion of conditions which might lead to it may not be amiss. These are:

(1) Excessive silica ratio, especially when accompanied by imperfect reduction, say too low a fuel ratio.

(2) Deficiency in number of bases, as for instance an ore containing no bases except iron, and a limestone with lime only as base. This assumes absence of alumina, or an exceedingly low percentage of same in ore and fuel.

The first condition was sometimes found in the old-fashioned charcoal practice. It was a pet fallacy of many furnace men in former days that to increase the lime burden was to increase the consumption of fuel, and the cost of charcoal was by far the most important item in expense account. The reasoning, though fallacious, was simple, viz.: "The more you put into the furnace, the more you have to melt." This, to the uninstructed "boss" seemed axiomatic. That it might take more fuel to melt 100 lb. of refractory stuff than 150 of a fusible slag, probably never occurred to him. Some of the silica ratios in these furnaces were terrifying, and it is not surprising that the slags were often "deeply, darkly, beautifully green," or that they had to be mechanically pulled out of the "notch."

Lime silica is a highly infusible product. But aside from questions of temperature a chemical influence comes into play. Silica tends to form polybasic silicates, the general rule being that increase in the number of bases produces greater fusibility. Whether, with high temperature and powerful reduction an iron furnace could be made to yield a pure lime silicate slag, and put all the iron into the metallic state, may be an open question. I greatly doubt it. Certainly a case can be cited in which there was a ruinous percentage of iron in the slag, which was rectified by the addition of aluminous ore to the mix.

Broadly, the iron furnace slag (manganese aside) is usually composed of silica, alumina and lime, or in the case of magnesian limestone of these constituents plus magnesia. MgO (see above) is dangerous only when alumina is high. With alumina not much over 10 per cent, there is no danger. With MgO and Al_2O_3 both about 20 per cent, the slag may become viscous.

We here give two actual furnace slags, both typical of everyday running. Both are averages, but the variations from week to week were trivial.

	I.	II.
Silica	39.54	38.25 per cent.
Alumina	16.82	7.85 " "
Lime	36.94	41.96 " "
Magnesia	6.58	11.84 " "
	99.88	99.90

If the reader will take the trouble to "formulate" these slags as silicates he will find that they are each astonishingly near to perfect "singulo" formulæ. They have in fact been selected for that reason. But if we compare them on the basis of "$SiO_2 + Al_2O_3$" as acid, they differentiate.

It is the usage at furnaces using Al_2O_3 as an acid component, to bring their slags between the limits:

$$\text{acid} : \text{base} = 1 : 1.5$$

and

$$\text{acid} : \text{base} = 1 : 1$$

that is, the minimum of silica-plus-alumina should be 40 per cent, and the maximum 50 per cent. Usually, about 45 "acid" to 55 "base" or a ratio not far from 1 : 1.2, indicates average practice.

Our No. II conforms, as its ratio of "acid" to base is as 1 to 1.17 or 46.10 to 53.80 in percentages.

But No. I gives:

$SiO_2 + Al_2O_3$	56.36 per cent.
Bases	43.52 per cent.

Ratio of acid to base is as 1 : 0.77. It cannot be said that No. I was as free flowing as No. II, but it was accepted as a good working slag.

The illustration serves to show that a far wider range is permissible for the ratio of the composite "acid," than for "acid" considered as silica alone. In fact the "oxygen ratio" is no longer much considered in the iron furnace.

"It is found in practice that the efficiency of slag varies more nearly with the basicity than with changes in the oxygen ratio." (Forsythe.)

Copper.—Slags from the copper furnace (matting) show wide variations, as it is usually impossible to make a slag of exact pre-arranged formula or percentage. In pyritic smelting there is a large margin, if only because we cannot always predict the amount of oxidation of the iron, (i. e., from FeS_2). According to the oxidation will be the ratio of slag to matte.

In stating as below the allowable variations in constituents of copper slags it should be remembered that manganese may be classed as iron, and barium and magnesium as calcium.

Slag constituents.	Minimum.	Maximum.
SiO_2	26	45 per cent.
Al_2O_3	0	20 " "
FeO (MnO)	18	65 " "
CaO (Mg, Ba)	0	28 " "
ZnO	0	14 " "

Lead.—The following table of lead slags, showing percentage only, not formulæ, is taken, as is the above maximum and minimum statement for copper slags, from the late Howard van F. Furman's "Manual of Practical Assaying."

TYPICAL LEAD SLAGS.

No.	SiO_2	FeO	CaO	ZnO
1	35	28	28	..
2	34	34	24	..
3	34	34	17	7
4	30	40	20	..
5	30	48	12	..
6	28 30	54	6	..

These do not sum to 100, an apparent anomaly to be later explained.

We shall use the "excess" method at first, and in addition

to fluxing problems give some illustrations in the application of stoichiometric principles.

(a) Ore has: silica 40 per cent, alumina 20 per cent, as its only "slag" constituents. Calculate for slag on a "singulo" silicate basis (alumina as a base) having a limestone containing silica 10 per cent and calcium carbonate 90 per cent. Calculate on 100 lb. of ore.

Solution.—First find SiO_2 excess in the ore itself on the required basis. Alumina singulo silicate is $2\,Al_2O_3 + 3\,SiO_2$, stoichiometrically 204 : 180.

$$204 : 180 = 20 : 17.64 = SiO_2 \text{ required for } Al_2O_3.$$
$$40 - 17.64 = 22.36 = SiO_2 \text{ excess in the ore.}$$

Next find CaO excess in the limestone. Enough CaO must be taken out of the limestone *to satisfy its own silica,* before we can compute for the ore. In other words, we must get the *lime excess* of the limestone and compare it with the *silica excess* of the ore. Under the given condition, the excess calculation must be for the singulo form:

$$60 : 112 = 10 : 18.66.$$

This 18.66 is lime (formulistically $2\,CaO$) required for the silica of the limestone.

Total lime in the limestone is $90 \times 0.56 = 50.4$ per cent. Then, $50.4 - 18.66 = 31.74$ = excess of CaO or "available" lime.

The problem is now reduced to this, viz: "An ore with 22.36 per cent of silica is to be adjudicated as to a lime singulo silicate, with a limestone containing 31.74 per cent of CaO." For the alumina of the ore is out of the calculation, having been set aside with its proper amount of silica in the first operation.

Also, the silica of the limestone is out of this final computation, having been set aside with its calculated percentage of lime.

These quantities, being in their proper singulo ratios, do not in the least disturb the calculation of the remaining portion of the materials, nor do they affect the correctness of the outcome, since that must be in "singulo" form.

Thus we have now to do only with the excess SiO_2 on the one side, and the excess of CaO on the other. These are often called, as above, "available" quantities.

Our proportion will be:

$$SiO_2 : 2\,CaO = \text{excess silica} : \text{required CaO}$$
$$60 : 112 = 22.36 : 41.74$$

The only remaining question is: "How many pounds limestone containing 31.74 per cent available CaO will furnish 41.74 lb. CaO?"

$$31.74 : 100 = 41.74 : 131.5$$

It will then require 131.5 *lb. of the limestone* to satisfy 199 lb. of the ore.

This as an initial type excess problem, has been detailed at great length. In future certain steps, i.e., those always involved in similar cases will be taken for granted.

(b) What will be the percentage analysis of the slag from problem (a)? Assemble the constituents from 100 lb. of ore and 131.5 lb. limestone.

	SiO_2	Al_2O_3	CaO
From 100 lb. ore.....	40.00	20.00	.....
From 131.5 lb. limestone.....	13.15	.....	66.27
	53.15	20.00	66.27

Weight of slag for 100 lb. ore = 139.42 lb. Analysis:

$$SiO_2, \frac{5315}{139.42} = 38.12 \text{ per cent}$$

$$Al_2O_3, \frac{2000}{139.42} = 14.34 \text{ per cent}$$

$$CaO, \frac{6627}{139.42} = 47.53 \text{ per cent}$$

99.99 per cent

Arithmetical Excess.

In the above example we have assumed that "excess" meant excess above a given chemical ratio, but it is very frequently used to denote a difference between acid and base derived by simple subtraction without any reference to chemical adjustment. In fact, if the requirement for the slag is purely numerical, the handling of the "excess" is reduced to the same basis. In iron furnace practice the requirement is sometimes no more than ratio of "acid" to "base," which greatly simplifies the problem.

In the following problem we adopt what has become a common usage. The "oxygen ratio" is discarded entirely, and the sum of SiO_2 and Al_2O_3 is considered as the "acid" side of the slag. All other bases are simply added together without any regard to chemical ratio. An arithmetical ratio, acid to base, as stated above, from 1:1.5 to 1:1 is assumed. In the present case we use an intermediate figure.

(c) Analyses of the materials.

	Ore.	Coke.	Limestone.
SiO_2	4 per cent	4 per cent	5 per cent
Al_2O_3	2 per cent	1 per cent	2 per cent
MgO	4 per cent	.....	2 per cent
CaO	.....	.....	50 per cent
Fe_2O_3	90 per cent	.....	.....

Consider $SiO_2 + Al_2O_3$ as "acid"; $CaO + MgO$ as "base."

Requirement.—Slag to contain 45 per cent acid and 55 per cent base. This being very close to the ratio of 10:12, let us adopt that ratio as the basis of calculation.

Figure separately on ore and coke, as usual at iron works, i.e., state what "stone" is required for 100 lb. of ore and for 100 lb. coke. The reason for the usage is found in the variant fuel ratio of the furnace. By having a stated weight of "stone" for ore and fuel we avoid the necessity of recalculation for a change in the fuel ratio.

Preparation of the Data.—In the *ore* the only "base" is MgO, 4 per cent. Acid is $4 + 2 = 6$ per cent.

$$12 : 10 = 4 : 3.33 = \text{acid required by MgO}$$
$$6 - 3.33 = 2.67 = \text{acid excess in the ore.}$$

Coke.—Acid excess is $4 + 1 = 5$.

Limestone.—Base 52 per cent, acid 7 per cent.

$$10 : 12 = 7 : 8.4 = \text{base required by its } SiO_2$$
$$52 = 8.4 = 43.6 = \text{basic excess.}$$

This completes the preliminary calculations. We are now to find:

(1) "Stone" required by the ore:

$$10 : 12 = 2.67 : 3.2 = \text{required base.}$$
$$43.6 : 100 = 3.2 : 7.34 = \text{required limestone.}$$

(2) "Stone" required by the coke.

$$10 : 12 = 5 : 6 = \text{base required.}$$
$$43.6 : 100 = 6 : 13.67 = \textit{stone required.}$$

Proof.—Let us assume any fuel ratio, for illustration, and then compute total slag. If the latter proves to have the composition:

$$\text{acid} : \text{base} = 10 : 12$$

the proof is complete. Let ore to charge = 1000 lb., coke = 700 lb.

	$Al_2O_3 + SiO_2$	$CaO + MgO$
In 1000 lb. ore.....	60.00 per cent	40.00 per cent
In 73.4 lb. "stone"..	5.14 per cent	38.17 per cent
In 700 lb. coke.....	35.00 per cent	.....
In 96.32 lb. stone....	6.74 per cent	50.09 per cent
	106.88 per cent	128.26 per cent

$$106.88 : 128.26 = 10 : 12.$$

In the above proof table the figures derived from the stone are simply products of the pounds "required" by the respective percentages as per analysis. For instance, 100 lb. coke required 13.76 lb. stone, then 700 lb. requires $13.76 \times 7 = 96.32$, and this multiplied by 0.7 and 0.52 respectively gives 6.74 and 50.09.

In order to simplify the above example, nothing was said as to subtraction of silica to provide for silicon in the pig metal. This, which is a very simple matter, will be shown in the next problem. In this we return to the "formulistic' requirement.

(d) Analyses of ore, stone and fuel.

	Iron Ore.	Stone.	Coke.
SiO_2	8.00 per cent	6.00 per cent	8.00 per cent
Al_2O_3	5.00 per cent	.....	.....
MgO	.....	6.00 per cent	.....
CaO	.....	46.00 per cent	.....
Fe_2O_3	87.00 per cent	.....	.....

Assume ore 100 lb, coke 70 lb. Figure for the stone required to make a singulo slag. Alumina figured as a base.

These are slightly simplified analyses from an actual case. The coke ash was not pure silica, but is so assumed to remove some complication from the problem.

The subtraction of silica from the ore prior to computing the slag is, as regards its amount, a somewhat arbitrary proceeding, as the per cent of silicon in the pig metal varies according to the heat and reducing power of the furnace. We shall in this case assume that 3 per cent of the silica will be reduced to silicon, that is, we shall figure on 5 per cent of silica in the ore instead of 8 per cent, leaving silica in the coke and stone as in the analyses.

The singulo formulæ are $2\ Al_2O_3,\ 3\ SiO_2$; $2\ CaO,\ SiO_2$ = and $2\ MgO,\ SiO_2$.

Under the conditions no set ratio between the bases is possible. As we have taken 100 lb. of ore, figures will indicate either pounds or percentages.

Use the formula in all the proportions and *nearest whole numbers* for the atomic weights. z z

(1) Take out alumina silicate from the ore.

$$2\ Al_2O_3 : 3\ SiO_2 = 204 : 180\ (= 17 : 15) = 5 : 4.412$$

Pounds SiO_2 required by alumina = 4.412. Sum up all the SiO_2 in ore and coke, and subtract 4.412.

SiO_2 in ore (8 — 3).....	5 lb.
SiO_2 in coke.....	5.6 lb. (8 per cent of 70)
SiO_2 in ore and coke.....	10.6 lb.
Less required by Al_2O_3.....	4.412
SiO_2 available for stone.....	6.188

(2) Find excess of CaO in the stone. We have 6 per cent MgO in the stone;

$$2\ MgO : SiO_2 = 80 : 60\ (= 4 : 3) = 6 : 4.5$$

That is, the MgO requires 4.5 per cent SiO_2. Silica in the stone is 6 per cent; subtract the 4.5, remains 1.5 per cent SiO_2 to be set against CaO. In order to bring CaO into the last term put SiO_2 first.

$$SiO_2 : 2\ CaO = 60 : 112\ (= 15 : 28) = 1.5 : 2.8$$

We might also have figured MgO and CaO in one operation by using factor $1.4 \times$ MgO.

(If it had happened that there was *not enough* SiO_2 to satisfy the MgO, the excess of MgO would have been added to the available (total) lime.)

We now have 46 — 2.8 = 43.2; that is, our "stone" has 43.2 per cent *available* CaO.

(3) Now we bring together the ore and the limestone, with their respective excesses. Since we are figuring on 100 lb. of ore, and seeking required stone, we put the SiO_2 excess first in the proportion:

$$SiO_2 : 2\ CaO = 60 : 112 = 6.188 : 11.551$$

Available CaO called for is 11.551 lb., hence,

$$43.2 : 100 = 11.551 : 26.738 \text{ limestone.}$$

Answer.—100 lb. ore requires 26.738 lb. limestone.

The decimals have been carried far beyond any practical necessity.

It remains now to assemble our figures and check up. Analysis of the slag may then be figured.

(4) Take 100 lb. of the ore, 70 of coke and 26.738 of "stone"; figure out actual pounds of each constituent. As this will give us total weight of the slag, the latter may then be figured in percentage, i.e., its analysis may be stated in advance.

	SiO_2	Al_2O_3	CaO	MgO
In ore	5.00	5.00	.....	.....
In stone	1.604	....	12.30	1.604
In coke	5.60	....	.....	.....
	12.204	5.00	12.30	1.604

Weight of slag per 100 lb. ore charged = 31.108 lb. Analysis:

SiO_2	39.23 per cent
Al_2O_3	16.07 per cent
CaO	39.54 per cent
MgO	5.15 per cent
Total.....	99.99 per cent

We shall now give a problem in which the data require some simplification before computing. Before doing so we should point out that the substitution of one base for another by means of one of the "factors" already detailed in Article II *does not affect the excess as figured from the analysis.* This is a priori evident to anyone to whom the stoichiometric relations have become familiar, but we here give a case in proof.

We have a limestone, CaO, 39 per cent; MgO, 9 per cent. Since MgO : CaO = 40 : 56 = 1 : 1.4 we can say $9 \times 1.4 = 12.6$; call this CaO and add it to the actual CaO; 39 + 12.6 = 51.6.

Now suppose the stone contains in addition to these bases 10 per cent silica.

Figure on basis of a singulo, and find basic excess of the stone. First take it as it stands i.e., without the transformation of the MgO.

$2\ MgO : SiO_2 = 80 : 60\ (= 4 : 3) = 9 : 6.75 = SiO_2$ required for MgO 10 — 6.75 = 3.25 = SiO_2 left for CaO.

$SiO_2 : 2\ CaO = 60 : 112\ (= 15 : 28) = 3.25 : 6.07 =$ CaO for the SiO_2.

Total CaO less required by $SiO_2 = 39 - 6.07 = 32.93 =$ *net excess.*

Now make the change of MgO into CaO, by using factor as above, we get total CaO = 51.6. Figure excess CaO in the regular way:

$SiO_2 : 2\ CaO = 60 : 112 = 10 : 18.67 =$ CaO required by SiO_2.

51.6 — 18.67 = 32.93, *i.e., precisely the former figure.*

(e) In our slag problem preceded by simplification, suppose an ore to analyze as follows:

SiO_2	36.3 per cent
FeO	16.0 per cent
MnO	5.9 per cent
CaO	5.6 per cent
MgO	4.0 per cent
S	5.1 per cent
As	0.6 per cent
Pb	18.0 per cent
Cu	4.4 per cent
CO_2 and O.....	4.1 per cent
Total	100.0 per cent

The "flux" is an iron ore containing:

SiO.	4.00 per cent
FeO	86.00 per cent

(The analysis would, of course, show Fe_2O_3, but this is reduced by the ratio of 10 : 9 to FeO for slag computation.)

We are to produce a *bisilicate* slag after clearing of matte and adjusting bases. Use 100 lb. ore as unit weight. Find weight of the "flux" to be added. Find weights and analysis of matte and slag.

(1) Copper in the matte:

$Cu_2 : S = 4 : 1 = 4.4 : 1.1 = S$ for Cu_2S.

weight of $Cu_2S = 4.4 + 1.1 = 5.5$ lbs.

(2) S remaining is 5.1 — 1.1 = 4 per cent. Take out FeS to the extent that this 4 per cent permits:

S : FeO = 32 : 72 = 4 : 9 = FeO taken for matte.

16 − 9 = 7 = per cent (lbs.) FeO remaining (for slag calculation), FeS in the matter = 4 lbs. S + 7 lbs. Fe = 11 lbs. FeS.

(3) Take out As and Fe as Fe_5As (speiss):

As : 5FeO = 75 : 360 (= 5 : 24) = 0.6 : 2.88.

FeO finally remaining for slag is 7 − 2.88 = 4.12.

Note.—It is useless if the analysis states the iron as FeO to reduce this to metallic iron in calculating out S and As. Take the equivalent amounts (molecular weights) of the compounds given by the analysis. As this may not be at once evident, we step aside for a moment to explain.

Take out the As in the ordinary way, viz.:

As : Fe_5 = 75 : 280 = 0.6 : 2.24.

That is, metallic iron required = 2.24 per cent. But we would now have to convert this into FeO so as to subtract, viz.:

Fe : FeO = 56 : 72 = 2.24 : 2.88.

This 2.88 is precisely the figure already obtained directly.

This is very elementary; but it is by no means uncommon to see just this "round the corner" way of calculating through lack of familiarity with the language of symbols.

The speiss weights 0.6 + 2.24 lb = 2.84 lb.

To resume the calculation:

(4) Reduce the bases to two groups, i. e., FeO and MnO in the one, MgO and CaO in the other.

FeO : MnO = 72 : 71. We do not regard this ratio, but call it equality, i. e., we simply add residual FeO to the MnO : 4.12 + 5.9 = 10.02. This we shall enter as 10 per cent.

MgO is 4.4 × 1.4 = 5.6 = nominal CaO.

Total CaO = 5.6 + 5.6 = 11.2 per cent to be reckoned as CaO.

We have thus "prepared" the ore for calculation. Whatever flux is to be used can now be computed.

The Cu, S, As, are disposed of. The Fe has been accounted for to the extent rendered necessary by the residual S and the As. The lead, under the supposition of its complete reduction, is out of the computation. Basis are ready for adjustment to silica.

Remains for calculation:

SiO_2	36.3 per cent
FeO group	10.00 per cent
CaO group	11.2 per cent
Lb. to be reckoned in slag	57.5 per cent or lb.

These, however, are not altogether "actual" pounds, since the 4-lb. MgO has become 5.6 lb. CaO. These and the lb. of FeO are the *chemical equivalents of pounds* in the special relation into which they are now to be brought with fluxing material. As we have twice gone into the detail of similar adjustments, we here indicate operations in outline:

Ore. SiO_2 required by its own CaO..... 12.00 lb.
SiO_2 required by its own FeO..... 8.33 lb.
20.33 lb.

36.3 − 20.33 = 15.97 = "available" SiO_2.

Iron flux. FeO required by its own SiO_2 = 4.8 per cent leaves 81.2 FeO "available."

Final adjudication:

SiO_2 : FeO = 5 : 6 = 15.97 : 19.16 = FeO required by SiO_2 in the ore.

81.2 : 100 = 19.16 : 23.6 = *weight in lb. required of the "flux."*

The slag, leaving out of account the difference in weight caused by substitution of 5.6 nominal CaO for 4 lb. actual MgO, figures out as follows:

SiO_2 from ore	36.30		
SiO_2 from flux	0.94	Total SiO_2	37.24
FeO from ore	10.00		
FeO from flux	20.30	Total FeO	30.30
CaO from ore			11.20
Weight of slag per 100 lb. ore			78.74

Analysis of the slag:

SiO_2	47.3 per cent
FeO	38.5 per cent
CaO	14.2 per cent
	100.0

Matte and Speiss.

	Weight, lb.	Per cent.
Copper	4.40	22.75
Iron	9.24	47.77
Sulphur	5.10	26.37
Arsenic	0.60	3.10
	19.34	99.99

We have tacitly assumed that all the sulphur and all the arsenic would be found in the matte. Any pre-established rate of loss, as derived from experience, should, of course, be subtracted in the first place as a part of the "preparation" for computation.

The cases so far given which can be worked from either a formulistic or percentage bases by "excess" are of a very simple type. They regard in either case nothing but the ratio of silica (or "acid" $SiO_2 + Al_2O_3$) to base, and do not bring in any requirement as to the quantitative relations of basis among themselves.

Moreover, the complexity introduced by the use of two fluxes, usually an iron oxide and a limestone, has not been touched upon. As a general problem the requirement of a specific relation between the bases (or basic groups) cannot be met by the use of one ore and one flux.

Example. An ore, an "iron flux" and a limestone have the following analyses (slagging constituents only in the ore):

	Ore.	Iron flux.	Limestone.
SiO_2	40	10	5
FeO	10	80	..
Bases	5	..	....(CaO) 50

Suppose the slag requirement to be:

SiO_2	40 per cent
FeO	40 per cent
Bases	20 per cent

We see that the relation between silica and bases in the ore, though the FeO and base have been purposely set in the required ratio of 2 : 1, gives an excessive silica ratio. It is hardly necessary to demonstrate by figures that we cannot use either the ore and the "flux," or the ore and the limestone, and get a mixture for the required slag. If, by means of the ore and the iron flux we equalize SiO_2 and FeO, we drop the "base" far below its required figure, and it would be equally impossible to use the "stone" alone under the conditions of the problem.

Upon attempting the use of all three, calculating by the method of excess, we see that the solution is possible only by trial and readjustment.

Into this procedure we do not go, since by the method to be presented this and similar problems are solved directly and far more easily.

We shall, in fact, present a direct solution of the complex case:

	Ore.	Flux.	Limestone.
SiO_2	a	b	c
FeO group	d	e	f
CaO group	g	h	i

the case, namely, where all three of the materials on hand contain each and all of the constituents to be computed.

The computer in such a case, using "excess," would conclude that he had a problem on his hands "more curious than useful."

It is not impossible, but it is tedious and useless, to adopt a difficult method when an easy one is at hand.

Denver, Colorado.

Symposium on Research

A JOINT meeting of the New York Sections of the American Electrochemical Society, the American Chemical Society and the Society of Chemical Industry was held in the Chemists' Building, New York City, on Feb. 9. The meeting, which had been arranged by the New York Section of the American Electrochemical Society, was devoted to a symposium on research.

The chairman of the Section, Mr. Elmer A. Sperry, referred in his introductory remarks to the difference between accidental discovery and invention as the result of research and pointed out the industrial results of Faraday's research work. For research a true pioneer spirit is necessary—the desire, coupled with inspiration, to tackle something that is difficult, something that seems impossible. Nothing else is worth while.

Five papers were then presented by Messrs. Lidbury, Comey, Bennie, Lindsay, and Coolidge, which are given here almost in full.

Research in the Small Factory.

Research as a Gamble

Mr. F. A. Lidbury, of the Oldbury Electrochemical Company, of Niagara Falls, N. Y., pointed out that in spite of the modern tendency to the aggregation of industrial processes into units and combinations of enormous size, it still remains a fact that a very considerable proportion of industrial operations are conducted by organizations which are small.

The admiration of the modern large industrial organizations has led to attempts to model such small organizations as miniature imitations of the large ones without due consideration being paid to the fact that the needs of the two classes may be and almost always are entirely different. To mention but one instance, whereas in a large organization it is possible to draw up a perfectly definite staff chart in which duties of a specialized character are assigned to each member, in a small organization it is usually the case that duties of a very diversified nature have to be assigned to one and the same individual.

"In such cases there can be no question of a certain position carrying with it perpetually absolutely definite duties, for it may not be possible to find an individual whose training and abilities would qualify him for the exact diversified duties of the man whom he replaces; this necessitates a much looser and more fluid form of organization in which the important question is how best to divide up all the duties to be performed among the staff; and it is clear that a change of personel will frequently involve a change in the distribution of responsibilities and duties.

"Now it will be found that this difference between the two types of organization extends even to the organization of research. Classical examples of the organization of large research departments are known to us, and several have been excellently described; but here, again, we usually have a system built up of highly specialized units. I am inclined to think that there is possibly too great a tendency to assume that the admirable and remarkably successful systems of such organizations ought to be imitated as far as possible when it comes to doing research work in a smaller aggregation, but if it is borne in mind that the essential feature of these systems is the high specialization of the unit, it is obvious that when we come down to cases in which research has to be conducted by two or three, or even one man, we are dealing with an entirely different kind of condition and too slavish imitation will lead us at once into serious difficulties.

"The suggestion I have to make is that as differentiation becomes a virtue in proportion to the size of an organization, so integration becomes a virtue in the smaller type of organizations. You cannot in such cases afford a large staff of research men, each equipped to fit his own special position in a large machine; you have instead, one, or perhaps two or three men, and in such cases it is the more necessary that their training be such as to broadly equip them for technical investigation. Bearing this in mind, it becomes obvious that the smaller the number of research men the more important it is to avoid anything like a sharp separation of the research department. Such men must be not only investigating chemists, they must be thoroughly familiar with the technique of commercial operations; they must have no little practical experience of chemical engineering; they must, above all, have rid themselves of the habit of thinking in test tubes and of the unthinking worship of laboratory accuracy and scientific efficiency, and be able even in their laboratory work to constantly bear in mind the conditions of its translation to a large scale and of the commercial as well as the scientific significance of their results. They must, in other words, have that type of experience which enables them to tell when, for instance, additional scientific efficiency is likely to make an increase rather than a decrease on the cost sheet. All this is not likely to be found in a man straight from college, nor is he ever likely to acquire it if immured in a sort of holy of holies called a 'research laboratory.'

"I am well aware that research can be regarded as a method, and that a man who has grasped that method by his scientific work at college ought, if given proper opportunities, to be able to apply it in technical work once he learns to realize that while the form of his equations is the same, some factors are different, and others have quite a different significance.

"Research may, indeed, be 'a habit of thought,' but, then, so is the faculty of mathematical analysis, and I have never learned of any well-authenticated case of a man 'sitting in' at a game of poker with only this faculty and without experience of the game and getting much satisfaction. Indeed, this analogy does not seem to me to be much too extravagant; commercial research is very much of a gamble in any case; you put in your money and you don't know what you are going to draw, if anything; and it certainly seems as though to entrust the work on which the progress of an industrial concern depends to a man insufficiently familiar with technical operations and the factors in the equation of economy, somewhat resembles staking a mathematical professor of presbyterian descent in a crowd of professional sharpers. His experiences may be interesting and varied; he might even be able to gather from them enough material for a very important paper 'on some remarkable divergencies from the theory of probabilities,' but it is highly improbable that the commercial efficiency of his labors would be very high.

"It will be perceived that in the large model research organization such considerations need not enter. Specialized as the individual members may be, the department, as a whole, possesses, or has available, the necessary machinery for co-ordination. The segregated research man has, on the contrary, to possess within himself nearly all the elements of such co-ordination. If he is kept, or keeps himself, out of touch with practice, what is he to do? He is up against a stone wall.

"Nor it is obvious why we cannot carry this line of argument to its logical conclusion. This involves the proposition that there should be no separate research department in the small factory; that research in such cases should be done by chemists to whom is assigned as a regular part of their duties the supervision of and the responsibility for some part or parts of the plant.

"This probably sounds extreme. Of course, it is not ideal, but then the organization of a small plant is always a compromise. The problem is to make it as successful a compromise as possible, and I firmly believe that two chemists devoting part of their time to supervision and part to research will produce results at least equal in quantity and exceeding in quality those which would be produced by the same two men if one were kept entirely on supervisory work, and the other entirely on research.

The disadvantages attaching to discontinuous research may be readily admitted, but if the load-factor of research so carried on is obviously low, its efficiency is remarkably high. The danger of segregated research is that so much of it, though scientifically interesting, is technically worthless and consequently practically waste of time; and just as the technical analyst learns to confine his work to the limit of accuracy necessary in each particular case, so it becomes second nature with the technically experienced research chemist to limit his inquiries in a corresponding way. The amount of time saved is just as startling as in the case of analytical work, and this more than counterbalances the disadvantages of the bad load-factor referred to above.

"I do not want to be construed as underestimating the value of purely scientific knowledge and training in these matters, but in academic circles at least the fact is too often lost sight of that in the majority of important technical discoveries, theory springs from practice, and the practice arises first empirically. A Baekeland, for instance, synthesises a new plastic, works out methods of manufacture and applications; then as an afterthought, and presumably to salve his scientific conscience, he investigates the formula and decides that the substance is oxybenzyl-methylene-glycolanhydride. An Acheson discovers and works out a method of incorporating lubricating graphite in liquids; five years afterwards he discovers that he is dealing with a colloidal suspension. These instances are purely typical.

"Complaint is frequently made that research does not get a chance in the small factory; that it is difficult to get it introduced, and that when introduced, the research man is apt to have an altogether bad and discouraging time of it. So far as these complaints are grounded, they are probably due to the evil results of segregation. For this the manufacturer is not to be blamed so much as the chemist, and, in particular, the trainer of chemists. In these matters, the manufacturer who does not know is pretty apt to do what he is advised, and the tendency among a certain type of professor to glorify research as a kind of nostrum of a very superior character quite in a class by itself and in no way to be confused or mixed with routine has its natural results not only on the manufacturer, but, unfortunately, also on the student, who, not infrequently, is given the impression that a factory is merely a hot-bed of inefficiency and that he can step right in and by the use of his scientific training produce immediate and miraculous results. This is not at all a habit of mind conducive to success.

"The student on coming into contact with technical work must discover sooner or later that there are usually good reasons for things as they are and that it is infinitely more difficult to carry out a piece of research to a successful technical and commercial conclusion than to carry out a piece of research to a successful scientific conclusion. If this were impressed upon him previously it might save him a few, at any rate, of the jolts he is bound to get in practice. The manufacturer, on the other hand, should be impressed not so much with research as a scientific cure-all, as with research as a habit of thought to be introduced in the most intimate relation to daily practice. As things are, whenever a small concern boasts that it conducts a *separate* research department, it may be pretty confidently guessed that that research department is not paying cent for cent on its running expenses.

"One more word in regard to the efficiency of research in such a factory. It is too often the case that money spent in salaries fails to bring in its due return in work on account of the insufficiency of facilities provided. No doubt much excellent work can be done with makeshift apparatus, and no doubt, again, if a good library is not available in a works references can be hunted up somewhere else.

"But in commercial research more than in any other, time is an important factor; and it is astonishing that in so many cases a concern will employ not one, but two or three investigating chemists at good salaries and then stint them in regard to the supply of books and special apparatus. This again, I imagine, is a case where the manufacturer is doing just as he is told. He has been impressed by the reiteration that money spent in investigation will bring in rich rewards, but his informer has neglected to impress upon him that inadequate or insufficient equipment in the laboratory or works library is just as destructive of efficiency as in the factory itself."

Certain Phases of Technical Chemical Research.

Organization of Research of the du Pont Powder Co.

DR. ARTHUR M. COMEY, chief of the Eastern Laboratory, at Chester, Pa., of the E. I. du Pont de Nemours Powder Company, presented a paper on "certain phases of technical chemical research."

Twenty-five years ago there was hardly a manufacturer who pretended to have any chemical control of works' operations, not to mention the question of research, and some of the most important purely chemical industries did not employ even a single trained chemist. "Of late years this condition has greatly changed. The causes for this great development in chemical research laboratories connected with technical operations are undoubtedly complex.

"One cause, however, which has contributed greatly to this change is the enormous increase in capitalization of various manufacturing industries, making possible the expenditure of considerable sums on laboratories, both in equipment and personnel, and they have found that the establishment of research laboratories on a large scale has been a profitable venture, where as a small laboratory employing one or two chemists in research work may prove far from a saving investment.

"The fact must not be overlooked that research laboratories will be maintained for any considerable length of time, as a part of the manufacturing cost of a chemical industry only when they show a real or supposed pecuniary profit. A laboratory may be maintained in isolated cases for a shorter or longer time when the results obtained therefrom do not pay, but the solid foundation for the continued maintenance of a research laboratory rests on the fact that its operation directly or indirectly increases the profit in the industry with which it is connected.

"Manufacturers have been discovering within the last few years that large research laboratories do increase their profits and hence such laboratories are being erected in increasing numbers.

"The problem for the small manufacturer is a different one, and many plans have been devised by which he may avail himself of the benefits to be derived from the investigation of problems connected with his industry, without the necessity of establishing a large research laboratory. The usual way to go about this has been for the manufacturer, when he has become convinced that a little research work would help his business, to employ a chemist, just graduated from college, and provide him with a wholly inadequate laboratory, for neither he nor the young chemist has usually any idea of what is necessary in the matter. He then sits back and waits for results.

"When, at the end of a few months, he not only has not profited several hundred per cent on his research investment, but probably sees no return at all, he is very apt to give the whole matter up and feel that research work in his business at

least is a failure. If he had employed a chemist who had already been trained in a large industrial research laboratory and had provided him with the necessary equipment and had enough confidence in him to wait a reasonable time, it is, in my opinion, most certain that results would have been obtained which would warrant the outlay. It is almost impossible, however, that the necessary confidence could be inspired by the ordinary graduate fresh from college.

"The situation has been handled recently in a different way by the establishment of industrial fellowships or scholarships in certain of our universities. This scheme has its advantages in removing the necessity of permanently equipping a laboratory, and is undoubtedly more economical as far as the first outlay is concerned. When a comparatively small definite problem is to be worked out, it may be the best solution, but there are many difficulties in the way of extensive use of this manner of industrial research. The difficulty of making sure that the manufacturer who establishes the fellowship should have the whole benefit of the work being not one of the least.

"Another serious difficulty in the establishment and continuation of a system of fellowships in a college under one head would arise if the number of these fellowships should become large. The manifold details of the operations involved in each industry which it is necessary for the investigator to understand before he can undertake intelligent research work in that industry would eventually become too overwhelming for any one person to grasp, but it is the plan, as I understand it, that the instructors in charge of the department of the college to which these fellowships are attached should oversee and direct the work of the men engaged in the actual research, for they are not supposed to be qualified to undertake the research without such direction."

Dr. Comey then gave an interesting description of the very highly developed organization of research work in the E. I. du Pont de Nemours Powder Company.

"The chemical work of this company is brought together under control of the chemical department. This department occupies the same important position in the company as that held by the engineering department, which has charge of all construction work carried out by the company, and all chemical operations of the company are controlled ultimately by the chemical department. The chemical director occupies a similar position in the organization of the company as that held by the director of the chemical work in the large German companies like the Badische Anilin & Soda Fabrik, for instance. This is unique in American industry.

"The work of the chemical department is carried on under three distinct divisions. The first division takes charge of those investigations which are made on problems which cannot well be studied in the laboratory. The direction of this work is centered in the main office of the company at Wilmington under one head chemist. In this division are experts on all the various manufacturing activities of the company, nitroglycerine, nitrocotton, sulphuric acid, nitric acid, etc. These are chemists who have usually been trained in our research laboratories and are engaged in making exhaustive studies on the efficiency of our works operations. These men have their headquarters at the main office of the company, but often spend long periods of time at the works themselves, where they are assisted by other chemists especially detailed for this work.

"Besides the chemists in this division there is a force of chemists who carry on the routine work of the works laboratories. Each one of our works is equipped with a works laboratory, whereby a close control of works operations is maintained. The chemists engaged in this work are under the control of the superintendents of the various works.

"There is a total of about 250 *trained* chemists in the employ of the du Pont company.

"A second division of the chemical department is known as the Experimental Station. This comprises a laboratory or rather a group of laboratories where research work is carried on on problems connected with the manufacture of smokeless powder and black powder; also, problems which are brought to the company from outside are studied here, and from the results obtained the company is in a position to gauge the true value of any process that may be offered them.

"The Eastern Laboratory forms the *third* division of the chemical organization, and at this laboratory only problems connected in some way with high explosives are investigated. Under the term "high explosives" are comprised all explosives except black powder and smokeless powder. The term 'dynamite' is generally applied to these explosives, but dynamite is a much more comprehensive term than is ordinarily understood by those not acquainted with the explosive industry. The du Pont company, for instance, manufactures at least fifteen different and distinct kinds of dynamite, and of each kind many grades, sometimes as many as ten, representing different strengths—but the point I wish to emphasize is that the work to which the Eastern Laboratory is devoted concerns not one of the greatest industries in the country, like steel or oil, but one of the lesser industries, and only a branch at that—but in spite of this, it has been found to be a distinctly profitable undertaking to maintain a research laboratory on a decidedly large scale.

"The laboratory organization at the present time is made up as follows: A director, and assistant director, and 25 trained chemists; by trained chemists I mean men who have taken a regular chemical course at a college or scientific school. Among these chemists are several who have taken the degree of Ph.D. here or in Germany. This makes, with the directors, 27 chemists. Besides these, we have a professional photographer, whose whole time is devoted to the work of the laboratory, and an office force of six men, including a chief clerk, file clerk, storeroom keepers and stenographers. In addition to the above force we have about 50 men on our payroll, making over 80 in all. This list includes three carpenters, three plumbers and pipefitters, etc., but the majority of these men are ordinary laborers, whom, however, we have trained in many cases to make routine tests, etc.

"All the above force is engaged directly or indirectly in the investigations which we are carrying on. The director and assistant director are responsible for the work turned out in the laboratory, but in order to get the best results it is our policy to place a large amount of the direct responsibility upon the chemist who is actually carrying on the research. I believe in this way the best results can be accomplished. The chemist becomes vitally interested in his work when he knows that *he* is responsible for his results, and in this way the net result to the laboratory is the combined effort of several brains rather than the thoughts of one brain, executed by another's hands. I believe it is of the utmost importance in order to obtain the best results to make a chemist feel that the work is *his* to carry out; that *he* is responsible for the results, and that *he* will receive credit for the results of his work. This cannot be effected if more or less minute directions are given for the details of a resarch.

"Naturally, it is imperative, under this policy, to have first-class men, but I believe that the best results can be obtained in a research laboratory only with first-class men. Men to whom cannot be trusted the development of a research should not be employed in a research laboratory to carry on researches.

"Now, I do not wish to convey the idea that as we have 25 chemists working on research work we have 25 independent workers or 25 lines of research carried out simultaneously. As a matter of fact, we have about 10 lines of research at the present time, and the men conducting these researches have a number of other chemists to help them in their work. The number of these chemists varies according to the conditions. In some cases it is not necessary for the chemist to have any aid outside of that from the workmen. In other cases there is a large amount of analytical and testing work to do in con-

nection with the research. One of the head research chemists, for instance, has four other chemists working for him. They are under his direction, and only indirectly responsible to the directors of the laboratory. These men, as I said, perform the necessary analytical work and tests connected with the research in question, and beside these chemists it is our policy to furnish all the laboring men that the research chemists can profitably employ in their work.

"This becomes especially important in certain lines of work, as, for instance, the development of a new process in the laboratory. In most cases of this kind we try the proposed process out on a purely laboratory scale with ordinary laboratory apparatus. If the first results are promising, we may set up a larger apparatus in the laboratory, and then proceed to erect a small experimental plant where the same kind of apparatus is used as is proposed for actual works operations, but on a smaller scale; for instance, experiments on the nitration of glycerine are first carried out in a small laboratory where 40 grams is used as a charge. Then the operation is repeated at our experimental nitroglycerine plant, where we make about 90 lb. of nitroglycerine to a charge, instead of many hundred pounds, which is the regular works' scale; but all the apparatus at our small plant is of the same material and practically the same model as that used for actual manufacture.

"This development of a process on a semi-manufacturing scale together with the nature of the investigations and their connection with explosives leads to the necessity of providing a large number of buildings for our work. We have at the present time 76 buildings, spread over about 50 acres of ground, in use in our work at the Eastern Laboratory. The majority of these buildings are of considerable size, and it is customary to carry on only one process or perhaps part of a process in any one building.

"The economy of the plan of developing processes to a manufacturing scale under laboratory control is obvious to anyone who has had to do with experimenting at a plant on works' scale. We use our workmen for running these small plants, under the supervision of the chemist who is developing that particular process, and these men often go over to the works when a manufacturing plant is erected for the completed process and are well trained for operating it.

"This plan also helps to keep us in close touch with works operations, which is a most essential condition in order to make the results of laboratory work most available.

"The younger chemists working on the research work under the older men are also being trained, and as the work increases, or our older men are promoted to other positions, they in turn become independent workers, *if* they are capable of doing such work, which we are able to judge during their term of training. This furnishes a very efficient method for procuring men best adapted to the work to be undertaken.

"Another incentive offered to our chemists to exert their faculties to the utmost is the form of bonuses for unusually meritorious work. This plan, by the way, extends to all employees of the company. Each year men who have accomplished some definite result, such as invention and development of a new process, or have accomplished results in any form which has brought valuable returns to the company, are awarded bonuses in the form of a certain number of shares of the common stock of the company, 5, 10, 15, 20 or more. This stock is not presented to the beneficiaries directly, but they receive the dividends on the same from the date of award. The stock is held by the company and a certain percentage is credited to it annually by a system based on the earnings of the company. By this system the stock is paid for in a few years, when it is turned over to the recipient without restriction. If he leaves the employ of the company in the meanwhile, the amount that has been credited at that time is paid to him, but if he should die the whole amount is paid to his heirs.

"The benefits of a system of this kind are obvious. It serves as an incentive to bring out all there is in a man and also makes him interested to remain with the company. We place great importance on this last point, that of making a man interested in remaining with the company, if he does satisfactory work. The value of a research chemist naturally increases with his accumulated knowledge. It is our policy, therefore, to continue increasing the salaries of our research chemists so that they are high enough to make a position of that nature sufficiently attractive. Too often the policy has been to make the laboratory a stepping stone to other positions to the manifest detriment of the laboratory.

"The general policy of placing a comparatively large amount of responsibility on the men who do the work and having them realize that their efforts are thoroughly appreciated, and will be correspondingly rewarded if results are obtained, has contributed to make the operations of the laboratory a decided success, for it has been calculated that the laboratory has saved the company about 300 per cent of the sum expended thereon in the last three years. This amount, however, does not represent the real saving effected by the laboratory. It is based wholly on the visible direct saving brought about in the operation of a few manufacturing processes, where the saving on materials used, due to increased efficiency, could be directly computed in dollars and cents. The indirect gain due to improvement in product, introduction of new products, etc., has been wholly left out of this calculation. I do not think it unfair to judge of the success of a commercial research laboratory by the pecuniary saving effected thereby. No corporation is going to maintain a laboratory for any considerable length of time if it is run at a dead loss.

"The ultimate criterion of the success of a commercial laboratory is whether it pays or not, and in this it differs from research work carried on for purely scientific purposes. This fact may not always be easy to determine, as the benefits are often indirect."

Dr. Comey concluded by expressing the conviction that while it is useless to expect results from a small laboratory with untrained men, yet it would be a profitable undertaking for every large corporation in this country engaged in industries connected closely with chemical problems to establish research laboratories on a large scale, where investigations connected with their manufacturing operations can be carried on with the assurance of all necessary funds.

The Research Laboratory Working Independently.

Individual Research and Officialized Competition.

Mr. P. McN. BENNIE, of the FitzGerald & Bennie Laboratories, Niagara Falls, N. Y., presented a paper on "the research laboratory working independently."

"Research, by its very nature, is a thing dependent, requiring men and means; it presupposes the existence of a place where work may be done, a laboratory.

"It is not necessarily always profitable, as measured by dollars and cents, although its results may be extremely useful. The money expenditure is not, and should not be considered, an investment, for while that which is learned may be potentially valuable, it is not always a quick or convertible asset.

"Therefore, the research laboratory, working independently, eventually produces a lengthening list of discontented clients, because the expenditure which they insist upon considering a sure-thing investment turns out to have been an unprofitable speculation. In that respect the haphazard client of the independent research laboratory is very hard to educate. The ray of sunshine in this is that occasionally it does find a client broad enough and with sufficient experience to appreciate a negative result—for the money it saves.

"With the ordinary analytical laboratory such is not the case. Every completed analysis is a result, disappointing sometimes to the client, but still a result for which the laboratory is not blamed. A physical laboratory may test a piece of steel, for example, and while the specimen may prove to have

undesirable properties, such as low tensile strength or fragility, it is still a result, and no sane man thinks of holding the laboratory responsible for the result.

"Happily all research laboratories do not have, in addition to their other problems, the problem of the disappointed client. Some of the best-known and most appreciated organizations are those that have but one client—themselves. I refer now to laboratories such as are maintained and supported by large companies, like the General Electric Company, the National Carbon Company, Norton Company, Du Pont Powder Company and the Generel Chemical Company. Magnificently equipped as to men, munificently supplied with means, they have the apparatus and ammunition, so to speak, for effective attack upon the unknown. Such is research. It is gratifying to be able to record that among these, with some of whom we have had the pleasure to co-operate, do we find our most satisfying and satisfied clients.

"For effective research both men and means are essential, but the greater of these is men. The research workers of the future must be drawn from the young men of to-day, and these must receive their training in institutions conducted to that end. The young person should be carefully guarded and guided by those competent to do so. Then, to make the training useful, opportunity must be given to apply the knowledge gained, to exercise in work that which has been acquired in preparation, to obtain experience, and thus to develop the ability to do research.

"In each of these respects, preparation and practice, there are influences tending to restrict the usefulness of the prospective research worker. I believe that some allusion to these matters is pertinent and justified, particularly as to the sociological aspect. It is sad to relate that many colleges and universities exhibit distinct signs of commercialism, openly and even boastfully. I have recently seen reports where great stress was laid upon the large increase in the number of students and faculty, as though numerical strength should be considered an index of the quality of instruction given.

"Only a few days ago a graduate of a prominent university in the East told me that during his several years' attendance he considered himself fortunate if he saw his professor once a month. Upon these rare occasions he was given a few brief minutes, with the intimation that the professor must return to his work. His work, if you please, consisted in carrying on important researches for a company, and for a consideration. Another instance is that of the same person holding a professorship in two widely separated institutions of learning, where it is obviously impossible to give but a portion of his time to each. In this case great pride is taken in the fact that the professor has been able to prevail upon manufacturing concerns to appropriate sums of money for the purpose of commercial research by the professor and students. What the operating staffs of the companies affected think of such intrusions is not mentioned. . . .

"This is all very well for the professors, but how about the students? Are they only a consideration of secondary importance? Commercialism of this sort is already visible in the work and attitude of students, and thus we have the spectacle of what should be a course of preparation at an institution of learning degenerating into a mere apprenticeship in money-making, the university becoming an institution of earning—a shop.

"Our student graduates; he enters upon the practice of the profession for which he thinks he has prepared. Let us assume that his course has been in mining and metallurgy. The first work that presents itself, because it is generally the easiest to get, is some assaying. But he finds himself in competition with this or that state-subsidized school of mines, where assays are done at a nominal charge, too low to be profitable to practitioners. If he undertakes research on some problem in mining or metallurgy, he may find himself in competition with this or that government-supported bureau.

"Or let us assume that he has been working upon some specific problem, and is ready to realize a profit, either in money or acquired reputation, from the time and effort thus spent, only to find that a government report has just been issued by Bureau 'X' covering similar work, 'copies 5 cents.'

"Then, there is the Colorado movement, petitioning for federal aid to mining. . . . The people are being educated to demand the creation of this or that bureau to spend the money of all the people for something most of them know nothing about, do not want, and have no use for, for the benefit of the few.

"And so our student finds himself narrowed in training and restricted in opportunity, engaged in an undignified struggle for substance and subsistence with his alma mater on one hand, and officialized competition on the other.

"Destroy individualism and you destroy effective research. Stop research and the progress of science practically comes to a standstill.

"Darwin, Tyndal, Huxley, and Spencer, a quartette of the grandest contemporaneous thinkers the world has known, were the products of individualism and independent thinking, as contradistinguished from the necessarily restricted thinking of a governmental bureau."

In conclusion Mr. Bennie quoted the words of one of them, from Sections 851 and 852 of Spencer's Principles of Sociology: "This general drift towards a form of society in which private activities of every kind, guided by individual wills, are to be replaced by public activities guided by governmental will must inevitably be made more rapid by recent organic changes, which further increase the powers of those who gain by public administrations and decrease the powers of those who lose by them.

"There seems no avoiding the conclusion that these conspiring causes must presently bring about that lapse of self-ownership into ownership by the community, which is partially implied by collectivism and completely by communism.

" . . . . To bring about the change it needs but gradually to extend State-regulation and restrain individual action. If the central administration and the multiplying local administrations go on adding function to function, if year after year more things are done by public agency, and fewer things left to be done by private agency, if the businesses of companies are one after another taken over by the State or the municipality, while the businesses of individuals are progressively trenched upon by official competitors, then, in no long time, the present voluntary industrial organization will have its place entirely usurped by a compulsory industrial organization. Eventually the brain worker will find that there are no places left save in one or other public department, while the hand-worker will find that there are none to employ him save public officials. And so will be established a state in which no man can do what he likes, but every man must do what he is told.

" . . . . An entire loss of freedom will thus be the fate of those who do not deserve the freedom they possess."

The Commercial Aspect of Research

Organization and Accounting of a Research Laboratory.

Dr. Charles F. Lindsay, of the Remington Arms-Union Metallic Cartridge Company, presented a paper on "the commercial aspect of research." He quoted a remark by President Remsen in an address at an alumni dinner of Johns Hopkins University to the effect that the Government, large privately endowed foundations, and the research laboratories of the large manufacturing concerns are hindering the advance of research work in the universities each in its own way, but all acting toward the same end. They all divert some of the best men from the scientific life of the universities.

But in so far as a university has succeeded in instilling the investigator habit into its students and staff, in just such

measure is it fulfilling its highest object and ideal, and if it be a generality that at the present time commercial research has robbed our colleges and universities of some of the best men, the compliment is to the university and its methods.

The main object of Dr. Lindsay's address was the economic and business-like management of a research laboratory. It is a general principle that there are no results of laboratory experience which are not in their due order absorbed into the province of technology and, of course, once in technology, if not before, they must be thought of commercially. A difference between older manufacturing and operating methods and those of the present day is in the haphazard way such facts became formerly of technical importance, while to-day this is not left to chance, but is brought into use through the research laboratory.

The only difference, then, between purely scientific research and commercial research is one of degree and not of method or quality. The commercial chemist feels the time element more strongly. Commercial interests must of necessity take the place of scientific interest as defining and devoteing the subject to be studied and worked on.

Every research laboratory engaging in commercial work, no matter for what reason it was founded, must be of service or fail on a business basis. Every such laboratory must pay or be a failure. "By paying I do not mean that each year must show a cash balance sheet in favor of the laboratory, but it must show such a balance when account is taken of the work done and service rendered for which perhaps the books of the company may show no adequate return. In other words, it must be regarded as paying in the eyes of owner or directors."

The amount of money and time to be spent in an investigation is a problem which comes before a laboratory director very frequently, and the proper decision of which is of the utmost importance to him and his work.

In a large manufacturing concern each and every department has its own special problems which are brought to the research laboratory by their head, each man dilating on the importance of his requirements. It is usually utterly impossible to tackle all of them at once or perhaps more than one at a time. Here comes often the necessity for a careful investigation by the head of the laboratory of the individual needs and the making of a just choice independent of interest of the problem for active work. It may be, and often is, the case that the direct responsibility of such choice is taken off the shoulders of the head of the laboratory, but it is also true that in a great majority of cases his recommendations are acted on and the real responsibility is his.

"At this time also must be formed an idea which usually can only be an educated guess, though in some cases can be accurately calculated—of the commercial importance of the investigations. If the work be a success, are the returns to be large or small? What will be the saving in factory practice if the change suggested can be worked out?" . . .

"How shall the work be done to do it most economically? And I use 'economically' here in not the sense of cheapest, but as applied to the whole problem of time, money, and ultimate gain to your firm. Will it pay to put all available force at work and endeavor to attack the problem in a variety of ways and get it over with or is it best to take your time, go slow, put a very small force, perhaps only one man and maybe only part of his time and let the work gradually develop. These are perhaps the two extremes. Each case must be considered separately as each new work comes along, but I believe no new study should be undertaken without just such preliminary deliberation.

"There are, of course, investigations so broad in their scope and of so far-reaching a consequence and of so great an importance to your firm that the actual amount of time and money to be spent may become not a question of how much to spend, but what is the most that can be put into the work judiciously. It is on such a point as this that experience is really the only guide. Take the case of the development and improvement in lighting by electrically heated incandescent filaments. I do not think that the question ever entered Dr. Whitney's mind or the thoughts of the officials of the General Electric Company as to how far they would go in such a study, but how most judiciously they should spend time and money. New methods of study had to be worked out, familiarity obtained with little-studied elements, and a tremendous amount of work had to be done before it was really known how best to attack such problems. This work necessarily had to be slow, because the way was unprepared, but as these preliminary problems were worked out, it became good policy to put more and more men on the work and money was sunk right and left in a tremendous piece of experimental investigation, of which the present tungsten or 'mazda' lamp of the General Electric Company is the result.

"There are manufacturing concerns in this country which are so large and whose products are so varied that as far as materials and the necessary apparatus for experimental or development work are concerned are almost self-contained." This is, for instance, the case at the works of the General Electric Company.

"But the larger number of us, I believe, find a real problem in determining at times what to do when you require apparatus, machinery or conveniences that your plant does not offer. Under such conditions you may often find that to do the work properly will require an expenditure of time and money beyond what might be called the probable value of the experiment. On the other hand, there is no doubt that in general such experiments as the above condition brings to your mind, can be performed cheaper outside, for instance, by a manufacturer of such machinery as you desire or by some one who has in use such machinery or apparatus. At times you are really between the horns of a dilemma. On one side there is the natural desire for secrecy until such experiments be carried to completion and, second, the wish to overcome an unnaturally high expense in its performance. I take it as a high compliment to the American manufacturer that you can usually bring such experiment to him and have it performed with accuracy and dispatch and with a consideration of the confidence placed in him."

Dr. Lindsay thought that a good deal of mutual help might be extended between different research laboratories in the loan of certain apparatus or that there might be a field for the Bureau of Industrial Research under Government control.

"Not only do I feel that accurate commercial accounting is necessary in a modern research laboratory covering the expenditures over a given period for salaries, labor, materials, etc., but that as far as possible, or as the importance of the subject demands, an accurate and detailed accounting of each and every separate price of experimental work should be kept. For in *no* other way that I know of is it possible to accurately proportion laboratory charges among the various departments of the plant or to assign any accurate development charge to a new article or process.

"Of course, I realize that the carrying of this process to the limit is, in general, unnecessary and involving useless expense and that it is usually quite sufficient to keep the charge on miscellaneous and small investigations collectively and that only segregation of charges need be made on the more important and expensive work.

"As explained before, an estimate of the commercial importance of a proposed investigation should be made before its start and at the same time an idea obtained of the extent and expensiveness of the investigation. On this data application should be made for an appropriation covering the cost of the proposed study. With this appropriation, which may be called plant order, manufacturing order, experimental order or plain appropriation, duly numbered, it is perfectly possible and easy in any well-accounted plant, by carefully charging all time and material against such number, not only to accurately

keep the total cost of the work, but the progress of it can be followed as closely as required. . . .

"In other words, let a piece of experimental work as far as the accounting is concerned follow the same routine that is generally followed in erecting a new building, buying new machinery, or in putting a production order through the plant."

As to size of staff, Dr. Lindsay suggested that, as a rule, the staff should not be increased on account of pressure on one particular piece of work alone, but only when the increased force is permanently necessary and justified. "The only exception to such a working policy is, in my opinion, during the period of the summer vacation when it may be good policy, not so much with an idea of any present help, but to keep in touch with and find out good men, to temporarily engage help at a nominal salary from among the summer workers."

Roentgen Ray Research.

The Use of Tungsten for the Target.

Dr. W. D. Coolidge, of the Research Laboratory of the General Electric Company, said that most phenomena which are now known about Roentgen rays were already described by Roentgen. But since his discovery, wonderful strides have been made in creating and extending their field of usefulness, though a great deal of fascinating work remains still to be done.

Dr. Coolidge then demonstrated concisely the essential features of a Roentgen ray tube. What is being done in a Roentgen ray tube is to first produce a stream of rapidly moving electric charges and then to bring them to rest with abrupt suddenness by the "target." This produces the Roentgen rays.

To produce a stream of rapidly moving electric charges, a vacuum is required. The cathode, which must not disintegrate, is invariably made of aluminium. Aluminium is very good for this purpose. The spherical shape is essential to direct the electrons in a stream to the target.

In the target we have a remarkable case of highest concentration of energy. Since the percentage efficiency of transforming the kinetic energy of the moving electric charges into Roentgen rays is always very low, most of this kinetic energy is transformed into heat. It is for this reason that the material of which the target is made should have as high a melting point and as low a vapor pressure as possible. Heretofore platinum has been generally used.

But now since tungsten has become a commercial metal through the development of the tungsten lamp, the advantages of tungsten over platinum for target construction have been recognized. These advantages are that tungsten has a higher melting point than platinum, has a lower vapor pressure at high temperature, and also has a better heat conductivity which permits the carrying off of the heat produced in the target. Finally the density of tungsten is sufficiently high to bring the electrons suddenly to rest. For all these reasons tungsten is now finding very much favor as material for target construction.

Dr. Coolidge then discussed in detail different methods of taking care of the heat produced in the target, and finally referred briefly to the utilization of Roentgen rays for practical purposes. Some excellent radiographs were shown. As the interpretation of radiographs is difficult, it is generally left to specialists. Stereoscopic pictures are now often used. Chemical stunts have also been performed with the aid of Roentgen rays.

Dr. Coolidge's paper was illustrated by a number of very interesting exhibits.

* * *

The discussion of the papers was opened by a few words of Dr. Edward Weston, the distinguished electrical and electrochemical pioneer.

Dr. L. H. Baekeland spoke at greater length on a certain phase of the tragedy of chemical research; the balance has made chemistry possible, but until we get rid of the balance, chemistry will remain stagnant. He then spoke of the life of Pasteur and his wonderfully successful research work done under absurd external conditions. As to the work done by our Government bureaus, Dr. Baekeland referred to a visit to the Bureau of Standards, the Bureau of Chemistry and other bureaus in Washington as an inspiration, as it showed underpaid scientists doing faithfully splendid work.

The meeting, which was exceedingly well attended, then adjourned.

Resuscitation From Electric Shock.

The first meeting of the Commission on Resuscitation from Electric Shock was held in the board room of the National Electric Light Association, New York, on Feb. 22. This commission was organized upon the initiative of the National Electric Light Association, and has for a purpose the study of electric shock and the preparation of a set of rules for first aid in case of electrical accident. The commission is composed of members of the American Medical Association, National Electric Light Association, and American Institute of Electrical Engineers, as follows:

Nominated by the American Medical Association: Dr. W. B. Cannon, professor of physiology, Harvard University, chairman; Dr. George W. Crile, professor of surgery, Western Reserve University; Dr. Yandell Henderson, professor of physiology, Yale University; Dr. S. J. Meltzer, Rockefeller Institute for Medical Research, New York; Mr. W. D. Weaver, editor *Electrical World*, secretary. *Nominated by National Electric Light Association:* Dr. E. A. Spitzka, professor of general anatomy, Jefferson Medical College; Mr. W. C. L. Eglin, electrical engineer Philadelphia Electric Company. *Nominated by American Institute of Electrical Engineers:* Dr. Elihu Thomson, electrician, General Electric Company; Dr. A. E. Kennelly, professor of electrical engineering, Harvard University.

At the meeting in New York City the medical members of the commission unanimously adocated the Schaefer, or prone, method as the best means in the hands of laymen for maintaining respiration in victims of electric shock, and the commission formally voted to recommend this method. (The Sylvester method, in which the patient is placed on his back, has heretofore been used principally in similar charts.) A chart is now being prepared which will give details of first aid in cases of electric accidents and will describe fully the method of applying artificial respiration.

The Sherardizing plant of the Ohio Brass Company, Mansfield, O., is described in that company's *O-B Bulletin*. The process consists in coating iron castings with zinc by placing the castings in drums with a quantity of zinc dust, sealing the drums and running them into furnaces, where they are rotated and heated at a suitable temperature. As is well known, the zinc dust will not melt at temperatures above the melting point of zinc, owing to a film of zinc compound which surrounds each particle of zinc. In touching the iron, however, this film seems to be broken and the zinc is deposited on the casting, forming a coating which is said to be superior to that formed in the ordinary process of galvanizing. When the castings have been heated for a sufficient length of time they are drawn from the drums, after cooling, and a selected number of pieces are tested by immersion in a solution of copper sulphate. The chemical action dissolves the zinc and precipitates copper. Ultimately the zinc will be totally dissolved at points, and the iron will be exposed. The standard test adopted is that the casting shall withstand four successive immersions of one minute each without having the iron show through the coating of zinc. Between each immersion the casting is brushed and washed in clear water.

The Granby Smelter.

BY GEORGE M. LEE.

The Granby Smelter, the largest copper smelter in the British Empire, is located on the north fork of the Kettle River, one mile from the city of Grand Forks, British Columbia. It is owned and operated by the Granby Consolidated Mining, Smelting & Power Company, Ltd. The present company is the outgrowth of the organization and consolidation of smaller companies, starting in 1901 with the combination of the Old Ironsides and the Knob Hill claims at Phoenix, B. C. The credit of organizing and financing is due to Mr. J. P. Graves, of Spokane, who is the vice-president and general manager of the company. Mr. George Martin Luther, of Boston, is the president.

The company now owns forty-three claims and fractions in the district surrounding Phoenix, comprising, in all, some 1050 acres, and the smelter site of 118 acres at Grand Forks.

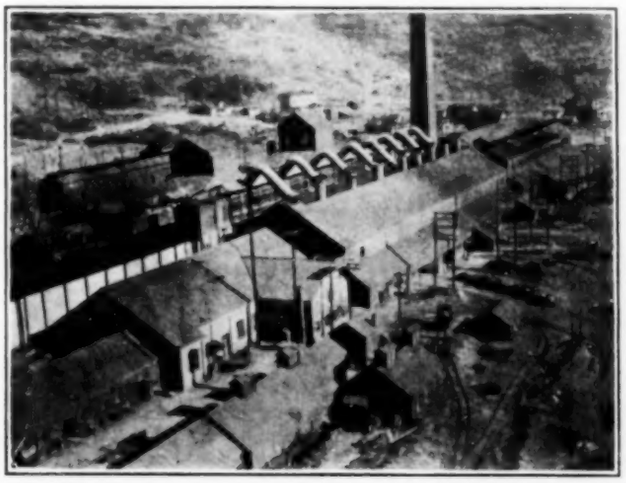

FIG. 1.—GENERAL VIEW OF SMELTER, SHOWING CONVERTER BUILDING, BLAST FURNACE BUILDING, SAMPLE MILLS, WITH BLOWER-ROOM AND SHOPS IN FOREGROUND.

Below is a statement of the financial operations of the company from 1904 to 1911:

The iron is present as silicate, uncombined oxide and sulphide. The lime and magnesia are present as carbonates.

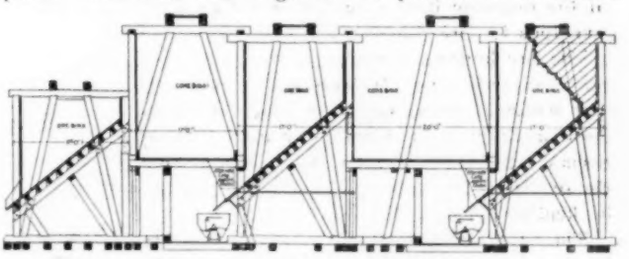

FIG. 2.—NEW ORE AND COKE BINS.

Chalcopyrite is the only copper-bearing mineral and the gold and silver value is carried in it. The ore is self-fluxing, which is a fact worthy of note, as otherwise the cost of smelting would advance materially. Since the organization of the company 7,299,911 tons of ore have been treated by the smelter and at present there are about 6,500,000 tons of ore blocked out in the mine. During the last year, moreover, the company developed more ore than was mined.

The company commenced the building of the smelter with the idea of adding units from time to time as the output of the mines was increased, and they have pursued that policy up to the present time, and as the smelter stands today, it shows a marked metallurgical and mechanical efficiency.

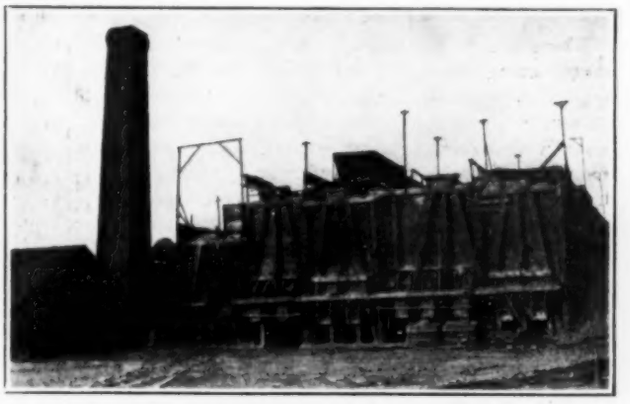

FIG. 3.—END OF BUNKERS.

GRANBY CONSOLIDATED.
FINANCIAL OPERATIONS 1904-1910.

Year	Gross	Expenses	Net	Shares	Earn. Per. Sh.	Pd. Per Share	Current Assets	Current Liabilities	Net Cash Assets
1904	$2,966,347	$2,682,883	$283,514	133,630	$2.12	$1.00	$376,075	$179,398	$196,677
1905	2,749,145	2,036,496	712,649	135,000	5.28	....	788,005	184,696	604,309
1906	4,751,059	2,927,441	1,823,617	135,000	13.51	6.00	1,256,598	103,903	1,152,695
1907	4,521,549	2,596,611	1,924,937	135,000	14.26	12.00	1,879,592	784,648	1,094,944
1908	3,790,184	3,183,662	606,522	135,000	4.50	5.00	1,614,962	898,219	716,743
1909	3,983,537	3,302,402	681,135	135,000	5.05	2.00	1,718,763	973,073	745,690
1910	4,099,925	3,534,978	564,947	148,496	3.80	2.00	2,239,197	178,776	2,060,421

	ORE TONNAGE Recovery per Ton				MATERIAL SMELTED			METAL PRODUCTION		
Yr. end. June 30:	Dry Tons Shipped	Lbs. Cop.	Oz. Sil.	Oz. Gold	Cost per Ton*	Granby	Foreign	Total	Ounces Gold	Ounces Silver	Pounds Copper
1901	172,258	31.49	.4406	.1003	$4.77	169,087	7,832	176,919	8,871	34,990	5,435,955
1902	206,162	27.23	.2952	.0808	4.08	293,645	7,455	301,100	30,786	274,511	10,836,851
1903	290,133	24.58	.2772	.0717	3.75	289,583	13,914	303,497	35,121	277,574	12,551,758
1904	514,387	22.87	.2619	.0608	3.35	516,059	40,472	556,531	54,493	275,935	16,020,986
1905	551,304	24.68	.2688	.0599	3.14	550,738	39,382	590,120	42,980	215,449	14,224,692
1906	796,528	24.30	.3107	.0513	2.87	796,188	36,158	832,346	50,020	316,947	19,939,004
1907	644,549	24.43	.3038	.0503	3.28	649,022	16,893	665,915	32,738	201,337	16,410,576
1908	865,030	23.42	.2865	.0454	3.11	858,432	24,179	882,611	40,068	300,204	21,092,288
1909	963,510	21.90	.2730	.0435	2.85	964,789	19,944	984,733	45,760	335,520	21,901,528
1910	1,178,853	18.70	.2281	.0370	2.50	1,175,548	21,829	1,197,377	48,752	356,746	22,754,899
Total	6,272,714	....	....	....	....	6,263,091	228,058	6,491,149	389,589	2,589,213	161,168,537

* Excluding marketing Blister Copper.

The ore is a low-grade copper ore with some value in gold and silver and is of the following average analysis:

	Per cent.
SiO_2	35
Fe	13
CaO	17
MgO	3
Al_2O_3	8

Receiving and Storing Ores.

When the original bunkers were planned the Canadian Pacific Railway afforded the only means of transportation between the mines and the smelter, and the capacity of the bunkers at that time was limited to 1000 tons. In 1906 the Great

Northern Railway Company extended its system to both mines and smelter, and the capacity of the receiving bunkers was then increased to 9000 tons for ore and 7000 tons for coke.

The ore bunkers are built in three sections, each section being 17 ft. wide and 770 ft. long. The bottoms are inclined at 45° and the lowest point is about 10 ft. from the ground. Partitions divide the bunkers into 14-ft. bins, so that the ore from the different stopes and drifts can be kept separately.

The coke bunkers are placed between the ore bunkers, as shown in Fig. 2. They are 20 ft. and 17 ft. wide, respectively, and are the same length as the ore bunkers. They have flat bottoms to increase the capacity and are not divided into bins. Both coke and ore bunkers are of heavy wooden construction. Underneath and running the length of the bunkers are three tracks for the charge cars. Two of these tracks are placed so that ore and coke can be charged into the same train. The third track serves one outside row of bins.

Sampling Mills.

There are two sampling mills, identical in construction, but differing slightly in their method of operation. One of these is used exclusively for custom ores and the other for Granby ores and converter slag. The first treatment in both mills is similar and is as follows: The ore is trammed in push cars from the bins to a gyratory crusher. The product from the gyratory, which will pass through a 3-in. ring, is elevated to the top of the mill and the first cut is taken. In the Granby mill this amounts to 20 per cent of the sample and in the custom mill it varies from 10 per cent up, according to the size of the shipment. In the Granby mill the reject passes to a storage bin, while the sample is reduced to ½ in. in a Blake crusher and another cut of 20 per cent is taken, the reject going to a storage bin as before. The sample then passes rolls set at ¼ in. and a third cut of 20 per cent is taken. This is quartered by hand and a sample of about 20 lb. is taken to the assay laboratory. For each sample four half-cars are taken at random each week from each of the three crushers at the mine. This gives three 70-ton samples, which amount is less than 1 per cent of the ore mined.

In the custom mill the first cut is saved in a storage bin, while the reject passes completely through the mill with the samplers idle. The reason for this procedure is that custom ores, which are mainly quartz ores, require crushing before they are used in the converters, and the sampling mill becomes, temporarily, a reduction mill. The sample is then put through the mill and a 20 per cent cut is taken after each reduction in size. The final sample from the rolls is split on a riffle, quartered by hand and sent to the laboratory. The capacity of each mill is 40 tons per hour and the storage capacity for crushed ores is about 5000 tons. Snyder samplers are used in both mills.

Blast Furnace Department.

The first unit of the blast furnace department, which was built during 1899 and 1900, consisted of two furnaces 44 by 160 in. hearth area and

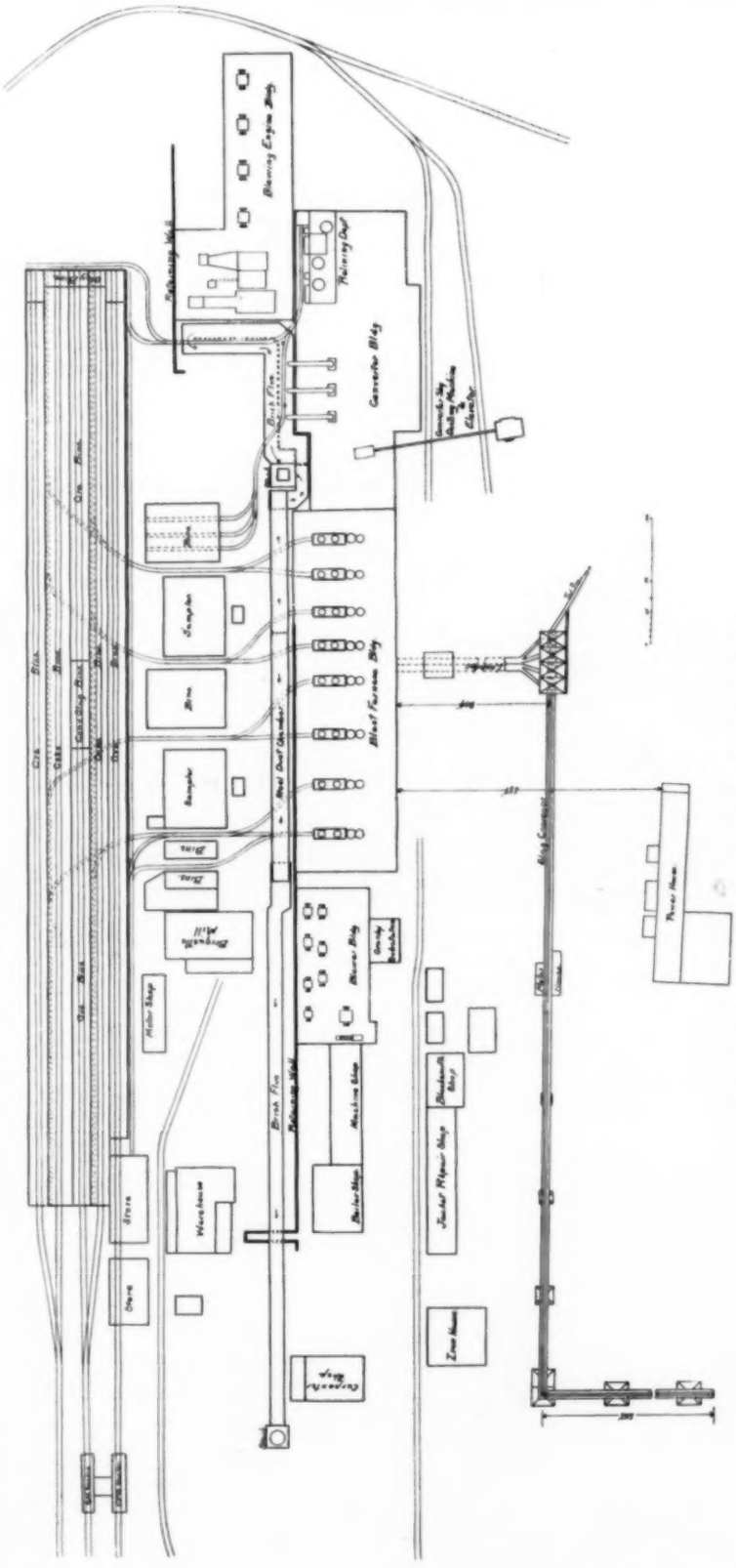

FIG. 4.—PLAN OF GRANBY SMELTER, GRAND FORKS, B. C.

having 5-in. tuyeres spaced 17¾ in. center to center. These made a 50 per cent matte, which was shipped to New York for refining. The slag was granulated and handled in launders.

Two years later two furnaces of the same design and size as the first were completed and put in operation, together with a converting plant of three small converters. In 1904 two more furnaces were added and a system of mechanical

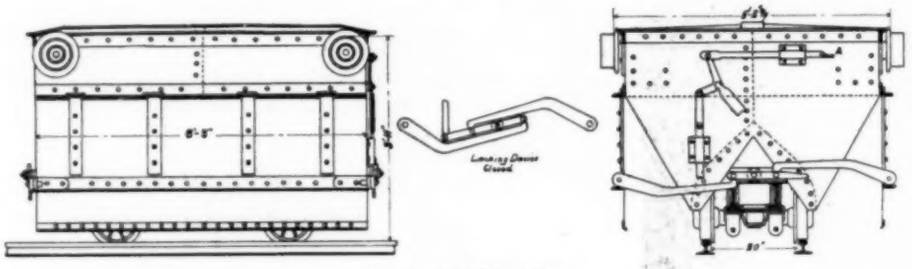

FIG. 5.—CHARGE CAR.

end charging was substituted for the system of side charging by hand which was in use up to that time. At this time two furnaces of a different design were planned to meet the requirements of an ore of higher sulphur content. In these the length of the furnaces was increased to 213 in. and the width at the tuyeres to 48 in. The tuyeres were 3½ in. in diameter and were spaced 8 11-16 in. from center to center. The lower jackets were straight line jackets.

In 1908 experiments under the direction of Mr. W. A. Williams, smelter superintendent, were made the basis for still further changes in design. All the furnaces were lengthened to 266½ in. and the column was deepened from 11 ft. 9½ in. to 15 ft. 8 in. Since then no further changes have been made.

The eight furnaces are housed in a steel building 290 ft. long and 71 ft. wide. They are in a row with their long axes at right angles to the length of the building and are spaced 27 and 41 ft. apart, as shown in the plan (Fig. 1).

The foundations are of concrete and the hearths are 2 ft. 4 in. above the level of the tapping floor. The jackets are of riveted steel, 7-16 in. thick on the inside and ⅜ in. thick on the outside. They extend from the hearth to the charge floor in two sections as shown. The lower jackets are 7½ ft. long, with a 4⅞-in. water space, while the upper are 7¾ ft. long and have a 3 5-16-in. water space. Water enters the upper jacket near the bottom and is deflected toward the bottom. It is drawn off at the top of the jacket and then circulates through the lower jacket in the same manner. This arrangement keeps the upper jackets cool and does not chill the tuyeres too much. The temperature of the water as it leaves the lower jacket is about 140° Fah. The eight furnaces require 2000 gallons per minute, or an equivalent of 3 gallons per minute per square foot of hearth area.

Air is delivered to the furnace through a 24-in. bustle-pipe connected with either one of two mains. The supply mains are 6 ft. in diameter and run lengthwise of the blast building behind the furnaces. Each furnace requires about 25,000 cu. ft. of air per minute and the blast pressure is from 24 to 28 ounces.

Above the charging floor the tops of the furnaces are built of red brick and have a fire brick lining. All brick work is reinforced and bound with stays and fifty 6-lb. rails. An opening is left in one end of the furnace for the entrance of the charge train and two hand holes are left in one side to allow for the tripping of the cars. No gate or curtain is used over the charging aperture.

The system of end charging was devised in 1904 by the then local manager, Mr. A. B. W. Hodges, and is worthy of comment for its efficiency, as well as for its uniqueness.

As shown by the sketch, in addition to the regular trucks, the charge cars have a wheel at each of the upper corners. When the car enters the furnace these wheels engage rails embedded in the side walls of the stack. At the same instant the lower trucks leave the ordinary rails which are discontinued at the entrance to the furnace, and the car is then suspended in the furnace. The gates of the cars are tripped by means of a rod inserted through the hand hole which engages the arm marked "A." The weight of the charge swings the gates and the charge is delivered toward the outside walls of the furnace. There is a slight tendency for the coarser particles to concentrate in the center and along

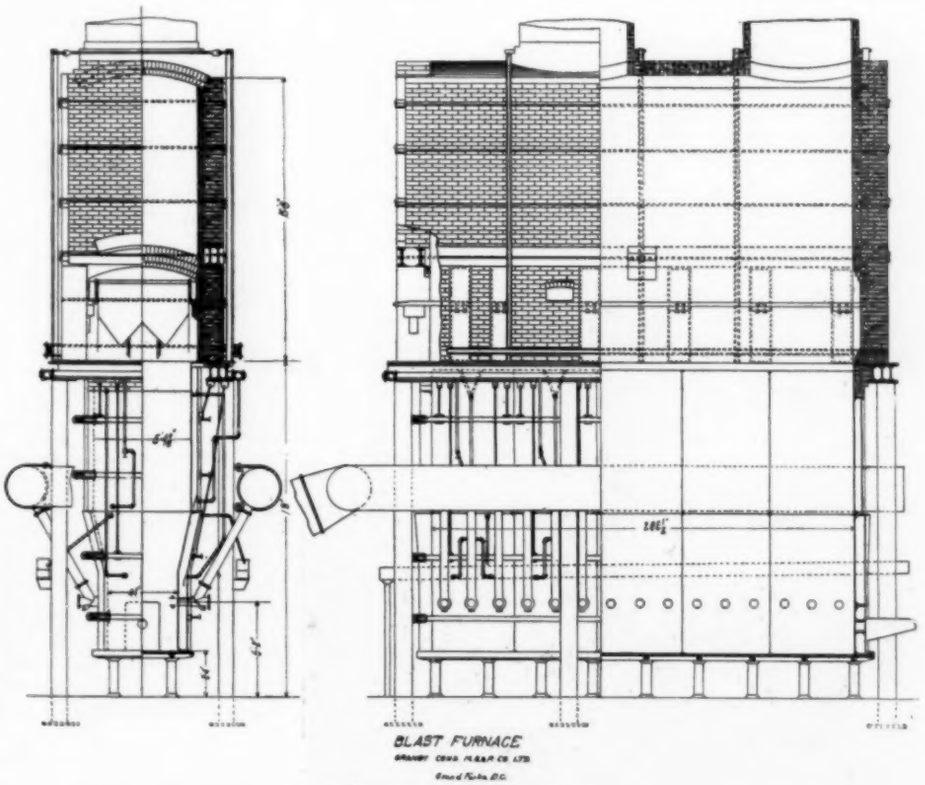

FIG. 6.—BLAST FURNACE DESIGN.

the outside, with the fine in the intermediate positions. This distribution gives an even descent of the charge. The time required from the instant the first car enters the furnace until it is withdrawn is, in actual practice, from 10 to 15 seconds.

Three cars making up the charging train hold about 10 tons of ore and are handled by 30-hp electric locomotives. Coke

and ore are charged separately and complete charges of both are made every 20 to 30 minutes. The charging cars are divided into four sections, and in case of uneven descent of the charge in the furnace ore can be distributed to the required places by omitting the charge from certain sections of the car.

The average daily capacity of each furnace is from 500 to

FIG. 7.—BLAST FURNACES, FROM TAPPING FLOOR, SHOWING SETTLERS.

600 tons of ore. A maximum for eight furnaces of 5000 tons has been reached, equal to 7¾ tons per square foot of hearth area in 24 hours. The coke amounts to about 12½ per cent of the ore charged. During the year ending June 30, 1910, the blast department smeltered the following tonnage:

	Tons.
Granby ore	1,175,548
Foreign ore	8,076
Flue dust	8,406
Converter slag and matte	52,678

The matte and slag discharge continuously from one end of the hearth into the first of a series of two settlers. The

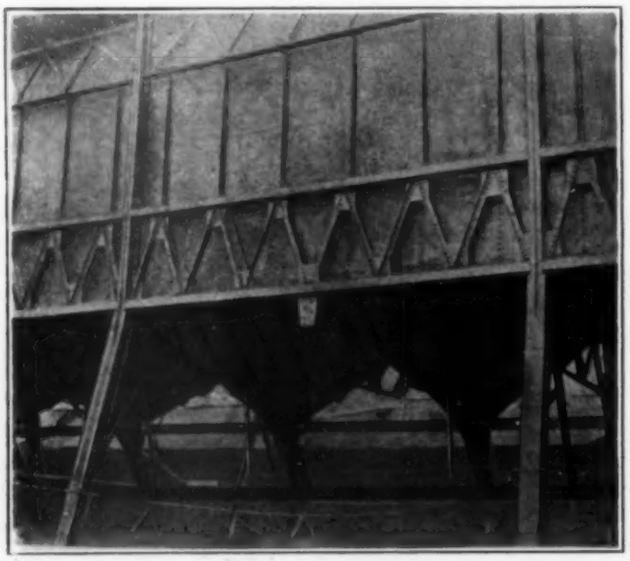

FIG. 8.—DISCHARGE HOPPERS ON BOTTOM OF STEEL DUST CHAMBERS.

first settler is 5 by 7 by 3 ft. deep and is water jacketed. It consists, primarily, of a ⅛-in. steel shell resting on a removable cast iron bed plate 2 in. thick. A course of fire brick is laid over the bottom and around the sides. The second settler is of the same general construction as the first, but is smaller, being 4½ by 5 by 3 ft. deep.

The first settler makes a practically complete separation. It is tapped as often as is necessary and the matte transferred to the converter building by a 10-ton electric crane. Any matte caught in the second settler is tapped once a day, cooled, broken and returned to the furnaces. Settlers are changed by means of the crane and the change requires 30 minutes.

Slag flows from the second settler into 5-ton slag pots mounted on trucks. These are handled, six or nine at a time, by 12-ton steam locomotives. These pots are dumped by hand, as they are in unstable equilibrium when full, and all that is required to dump them is the releasing of a catch. The slag analyzes approximately as follows:

	Per cent.
SiO_2	44.8
FeO	19.0
CaO	21.8
Al_2O_3	10.2
MgO	3.9
Cu	0.15 to 0.20
Au	Trace
Ag	Trace

All gases and flue dust are drawn from the furnaces into a chamber in which the greater part of the flue dust is col-

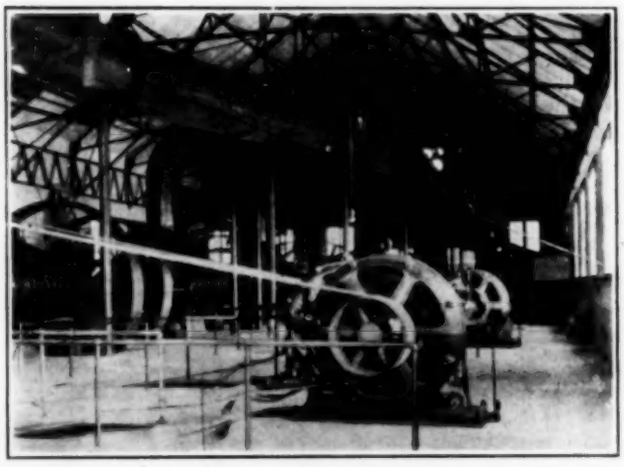

FIG. 9.—INTERIOR OF BLOWING ENGINE ROOM.

lected. This is a rectangular steel chamber 15 by 13 ft. in section and raised 21 ft. above the feed floor. It is parallel to the blast furnace house and at one end connects with a brick flue leading to a circular stack 13 ft. in diameter and 150 ft. high. The other end connects with a similar brick flue leading to a stack 12 ft. square and 153 ft. high. Flue dust caught in the steel chamber discharges from the hoppers on the bottom into a drag conveyor. This carries the dust to the briquetting mill, where it is moistened and briquetted in a White briquetting machine which has a capacity of 50 tons a day. All flue dust from the brick flues is cleaned up with shovels and wheeled to the mill in larrys. The briquetts are not dried before they are charged into the furnace.

Converter Plant.

When the smelter was first put in operation matte was shipped to New York for converting, but almost immediately the need for converting at the plant was recognized, and in 1902 the first converter building was erected. The equipment included three 72 by 100-in. stands and shells and the necessary copper molds and lining machinery. These converter stands were hydraulically operated.

As the number of blast furnaces was increased the converter equipment became inadequate and in 1909 these stands

were taken out and replaced by three 84 by 126-in. electrically controlled stands manufactured by the Power & Mining Machinery Company. Three 25-hp 220-volt direct-current motors furnish the power to operate these stands.

The quantity of matte charged varies from 3 to 9 tons, depending on the condition of the lining. The lining lasts usually for three charges, after which it must be renewed.

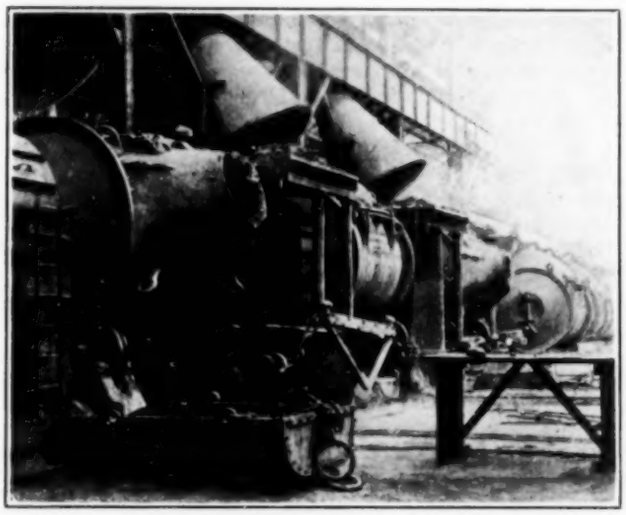

FIG. 10.—THE CONVERTERS, SKIMMING, BLOWING, POURING.

The full capacity of the department is 40,000,000 lb. of copper per year, but for the year ending June 30, 1910, the converters handled 31,868 tons of 37.5 per cent matte and produced 22,754,899 lb. of copper which averaged 98.5 per cent pure. The reason for the difference between the capacity of the plant and the real production is that the converters can handle in two shifts all the matte that the blast furnaces can produce in three shifts.

The copper is cast into 200-lb. bars and shipped to New York for refining. The molds are 24 by 33 in. in section and are mounted on 16-ft. trucks which run directly under the

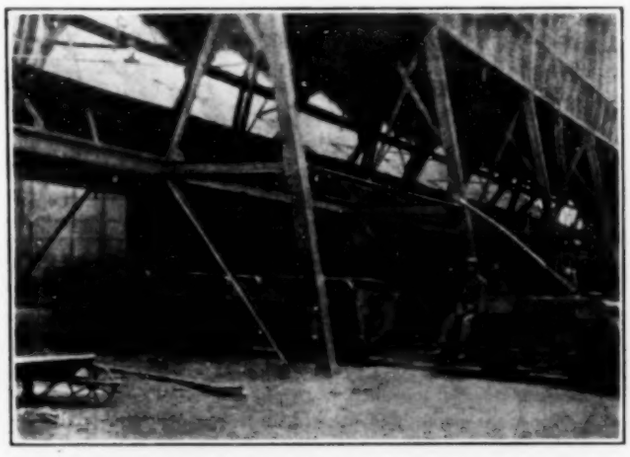

FIG. 11.—CHARGE TRAIN AND PORTION OF STEEL DUST CHAMBER

converters. These trucks are manipulated by cables and an electrically operated drum.

The slag from the converters, amounting to 100 tons a day, is cast in moulds, cooled, broken and dumped into cars to be returned to the blast furnaces. No mechanical means is used to break the slag, as the fall from the top of the conveyor to the car is sufficient to shatter the pigs. The analysis of the slag approximates 40 per cent silica, 55 per cent iron oxide and from 1 to 1½ per cent copper. The high percentage of iron oxide in the slag makes it valuable to add to a furnace which is running on a highly siliceous slag and also to furnaces that are working badly.

The converters are lined in the usual way with a course of magnesite or chrome brick next to the shell and then a 2-ft. layer of siliceous ore bound with clay. For the year ending June 30, 1910, 13,753 tons of siliceous ore and 4438 tons of clay were used in relining 1,573.5 converters. Fifteen hundred tons of coke were used in drying the converters.

The company has no difficulty in obtaining custom ores of high silica content, with little or no sulphur, iron or lime and containing sufficient value in gold, silver and copper to make them profitable. These ores come mainly from Republic, Wash. The clay comes from Hendrix Cut, Wash. The equipment of the relining department includes clay stalls, quartz bins and three mud mills. The whole department is served by a 40-ton electric crane with a 40-ft. span. A steel building 100 by 240 ft. in size houses the converter department.

Power Department.

Originally the company developed its own power from a power site on the north fork of the Kettle River, about 1 mile from the smelter. For this purpose a dam 25 ft. high, a flume and a powerhouse were erected and the machinery for developing 900 hp installed. At the end of the year 1902, when the plant was enlarged, this installation had become too small and instead of enlarging the power plant the company contracted

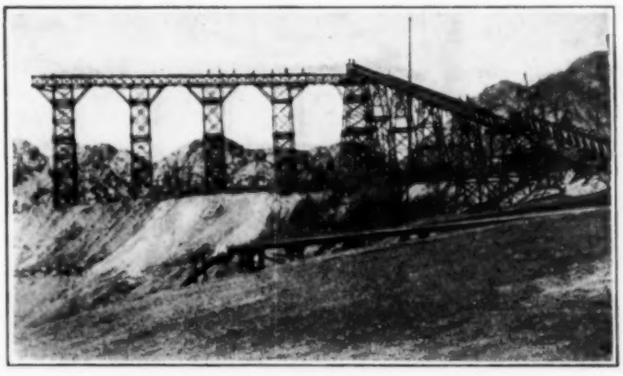

FIG. 12.—SLAG CONVEYOR DURING COURSE OF CONSTRUCTION.

for power from the Cascade Power Company operating on the Kettle River 10 miles below the smelter. In 1906 more power was required and 1200 hp was contracted for from the West Kootenai Power & Light Company. This power is transmitted 65 miles from Boonington Falls at 60,000 volts and is transformed at the smelter to 440 volts before use in the motors.

The Granby power house furnishes 110-volt direct-current for lighting and some 440-volt alternating current for use on the works. The generators are Westinghouse machines, direct-connected to water turbines. The power house also contains six pumps, each capable of delivering 500 gallons per minute against a 200-ft. head. Two of these are driven by water turbines and the other four are motor-driven. At the present time the company has just finished the installation of two more motor-driven pumps with a capacity of 1000 gallons per minute to furnish water for the new slag-granulation scheme.

The blower building has been recently remodeled, the old wooden structure being replaced with one of brick and steel. It contains three No. 7 and four No. 8 Connelsville blowers with capacities of 13,500 cubic feet of air per minute each. There is also one No. 10 blower capable of delivering 30,000 cubic feet per minute. The small blowers are separately driven by 100-hp, three-phase, 400-volt, 60-cycle, Westinghouse induction motors. The large blower is driven by two 150-hp motors

of similar design. There is also a small single stage compressor for furnishing air to the machine shop, at 90 lb. pressure.

The blowing engine room is also of brick and steel construction and contains four No. 10 blowers and three blowing

FIG. 13.—SLAG CONVEYOR DURING COURSE OF CONSTRUCTION.

engines. These are a Nordberg, with 36 by 36 in. cylinders delivering 3700 cubic feet per minute an Allis-Chalmers duplex, 34 by 34 by 36 in., delivering 6000 cubic feet per minute, and a Nordberg, 40 by 40 by 42 in., delivering 10,000 cubic feet. The first two are belt driven by motors of 200 hp and 300 hp respectively. The last is rope driven by a 500 hp motor. These engines supply air to the converters at a pressure of 12 lb. This building also contains a compressor for delivering air at 80 lb. pressure to the relining department, and a 200-hp motor-generator set for furnishing 250-volt current to the electric locomotives and the cranes.

There are also the machine shops, boiler shop, blacksmith shop and carpenter shop, each fully equipped for making almost any repairs. The smelter has just blown in after a four month's shut-down due to the strike of the coal miners in the Crow's Nest Pass and the consequent lack of coke, but as soon as the coke supply is adequate the smelter is prepared to run at the maximum capacity, as everything is in good repair.

Slag Disposal System.

Lack of dumping room within reasonable distance has made a change in the slag disposal system necessary, and the company has just finished the installation of a new system. Orig-

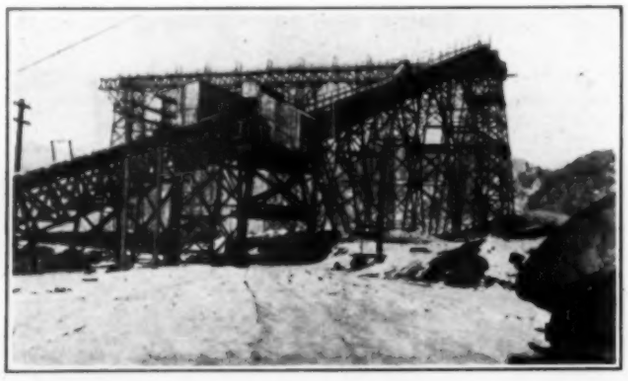

FIG. 14.—SLAG CONVEYOR COMPLETED.

inally the slag was granulated and disposed of in launders, but in a short time the available ground was filled to the highest level possible. So in 1903 the company installed a system of dumping molten slag from pots which were handled by steam locomotives. This system proved satisfactory and economical up until the last year, when the long haul made it expensive and slow. Now the company has gone back to the granulation scheme and plans to elevate the slag and deposit it on the top of the present dump, where there is room for some 5,000,000 tons.

The slag is granulated at the settler by an inclined jet of water from a ½ by 1¼ in. nozzle and falls into an inclined receiving trough of concrete. These troughs are about seven feet long and one foot wide, and connect with wooden launders which carry the slag to the receiving bin. The launders are lined with cast iron plates one inch thick and have a slope of ⅞ in. to the foot. The receiving bins are built of wood and have a capacity of 50 tons of dry slag, or a total of 200 tons for the four bins. From these bins the slag feeds on to a 24-inch belt and is elevated on a 18° incline, a vertical distance of 170 feet to a point 115 feet above the level of the present dump. It then discharges onto a horizontal belt running at right angles to the inclined belt. This belt is 252 feet long and an automatic tripper distributes the slag the length of the belt. When the dump reaches the level of the horizontal belt it will be moved laterally and the inclined belt extended.

The photographs accompanying this article will perhaps give a clearer idea of the smelter than the writer can convey in words.

Grand Forks, B. C.

Effect of Electric Current on Cyanide Solutions and a Simple Way for Their Preparation.*

By Dr. M. G. Weber.

As cyanide solutions give usually a brighter and denser deposit than acid baths and can be employed in almost every case, whereas acid solutions have but a limited field, they are in use in nearly every plating shop.

When one passes an electric current through solutions containing a double cyanide, for instance, double cyanide of copper and sodium, $CuNa(CN)_2$—I mention sodium and not potassium, as sodium cyanide is cheaper and more efficient than the potassium salt—this salt is separated into two parts, the cation Na and the anion $Cu(CN)_2$. If the current density is not too high and the copper salt is present in suitable concentration (about four ounces of metal per gallon) the cation will act with a reducing effect on the double cyanide of copper and sodium and turn out the copper on the cathode, according to the following equation:

$$CuCN\ NaCN + Na = Cu + 2NaCN.$$

We do not see, then any hydrogen gas developed, but free cyanide of sodium is formed. On the anode an oxidation takes place, the primary anion $Cu(CN)_2$ combines with the anode copper to the single cyanide:

$$Cu(CN)_2 + Cu = 2CuCN.$$

Cuprous cyanide is only soluble in cyanide solution, and the anode becomes covered with a dirty green mud offering quite a resistance to the current, unless free cyanide is present. We understand, therefore, that the cyanide being formed at the cathode is absolutely necessary to keep the bath in the proper working condition. In order to bring the free cyanide from the cathode quickly together with cuprous cyanide on the anode it is of advantage to stir the solution and also to heat it up, as in this way the diffusion will be considerably increased.

In case the current density on the cathode is too high, which can be perceived from an intense bubbling up of gas, the sodium no longer acts with a reducing effect, but is discharged and forms with the water of the bath sodium hydrate and hydrogen. The free cyanide is, therefore, lost, and this is the reason why cuprous cyanide on the anode cannot be entirely dissolved; a sludge will remain, dropping to the bottom of the tank.

*A paper presented before the Electroplaters' Association on February 10, 1912.

In order to bring this mud in solution, as is necessary to prevent the electrolyte from becoming impoverished in metal, sodium cyanide should be added. By using too high a current density on the anode molecular cupric cyanide is set free, which combines with water to basic cupric cyanide, forming a green mud. The reaction is as follows:

$$Cu(CN)_2 + H_2O = Cu< \begin{matrix} OH \\ CN \end{matrix} + HCN.$$

This hydrocyanic acid becomes polymerised and forms an insulating, cocoa-colored film which adheres to the anode. The addition of cyanide will bring this compound into solution.

In order to facilitate the dissolving of the anode metal and the secondary deposition at the cathode we add so-called conducting salts to the bath. If sodium sulphate, for instance, is used, the cation (Na) decomposes the copper sodium cyanide, copper being deposited, and simultaneously sodium cyanide is formed. The anion, e. g., SO_4, acts on the double cyanide in a known manner and cupric cyanide will result, which dissolves the anode metal.

It is interesting to note the conditions of voltage and current density which are required for cyanide solutions; for example, we take copper cyanide solution again. The ordinary voltage for a solution of this kind in a cold stage ranges between three and five volts. If, however, we use a hot bath, say from 170° to 180° F., in connection with a moderate current density and agitate it well we need no more than one volt. The current density for cold solution is about 3 to 4 amp. per square foot, whereas using hot, well stirred electrolyte we are able to go up to 20 to 30 amp. and still obtain a first-class deposit.

The preparation of cyanide baths according to older recipes, as, for example, that of Roseleur, is rather complicated. The chemicals must be pure and must be mixed carefully, in definite proportions and in a certain order of succession. It is difficult to be sure of the purity of the chemicals, and we all know how easy it is to make mistakes in weighing and mixing. Furthermore, one has to bear in mind that much time is lost in making the solution ready for use, especially for gold and silver plating, where most people start with the metal. The fumes given off in the preparation of solutions are disagreeable and dangerous.

All these difficulties and troubles are obviated by the use of some new products which are sold under the name "trisalytes." They combine in one salt all the necessary ingredients for plating baths; that is, an alkali cyanide, a cyanide of the metal to be deposited, and a conducting salt. The metal contents of these trisalytes are always constant, and in order to make up a plating bath it is only necessary to dissolve one of them in water. Voltage, current density and all other conditions remain just the same as in baths prepared after the old method.

By making baths with trisalytes fumes and poisonous gases are avoided, also there is absolutely no chlorine present, which has such a destructive corrosive action on the hooks on which the anode plates are fastened.

The trisalytes baths have a further advantage: that is, a longer life. They contain only the necessary ingredients required for electroplating, and no unnecessary salts are introduced, as, for instance, when copper or zinc carbonate are first converted into copper and zinc cyanide by the use of alkali cyanide with the formation of soda. It will, therefore, be easily understood why the baths can be readily revived by the addition of trisalytes.

With all these advantages the trisalytes are bound to facilitate, expedite and improve the work of the plater.

Perth Amboy, N. J.

The waste heat in slag has attracted the attention of many inventors who seek to convert it into steam and apply the latter to low pressure engines of the turbine type. Walter G. Perkins and William H. Barker have recently been granted a patent (996,132) based on this idea.

The Important Factors in Blast-Roasting.

By H. B. PULSIFER.

I.—Introduction.

The practice of blast-roasting which was inaugurated by the first Huntington-Heberlein process, originally of Pertusola, Italy, and first patented in the United States in 1898, has found so great favor with lead smelters that in some form or other it has now come to be part of the standard procedure at all large lead-smelting plants where a considerable proportion of the ore is available in the form of sulphides.

The success of the process is due to the fact that it is a good sulphur-eliminating process, that a great variety of ores are susceptible to the treatment, that the material is left in an ideal form for smelting in the blast furnace, and that the cost of the treatment is far less than the cost of any previously known method which would even approximately produce the same results.

The treatment of copper ores containing sulphur has been brought to considerable perfection as regards sulphur elimination in blast furnace, reverberatory and converter; this method is entirely inapplicable to lead ores in that exactly opposite conditions are necessary for the best work. In fact, should the uniformly quiet reduction and easy gas passage of a lead blast furnace be disturbed by local breakings through of the blast, such as might possibly represent the normal working of a copper blast furnace, the results are at once disastrous in that values are unduly lost by volatilization, the matte runs high in lead, and the slag will easily enough carry away twice the values it should.

On this acount any oxidizing influence whatever is precluded in the blast furnace, and every attempt is made to have the charge already fully prepared for the straight reduction when it reaches the furnace. Blast-roasting accomplishes the desulphurization and agglomerating in a way which leaves its product ideal for the blast furnace.

A close approximation to the original Huntington-Heberlein process still persists in certain localities where heavy galena ores are to be made ready for smelting; a modification involving pre-roasting but no addition of lime is widely used where pyrite ores form a considerable part of the sulphides. Further deviation pre-roasts the priming or igniting layer only, and finally, as at Midvale, Utah, no part of the charge is pre-roasted, but a fire on the grate ignites the mixed raw charge which is dropped upon the hot bed of burning fuel.

Although nominally considered of the best mechanical design and construction, the plants for this type of roasting have not always proven satisfactory in operation. Slight precedent and unknown demands may well account for the inability of the engineers to meet conditions where the costs of the mechanical operations and handling of materials have suddenly become the chief cost factor.

The introduction of the down-draft machines, at present represented only by the Dwight-Lloyd types, will unquestionably again make a general lowering of the cost of the roasting and at the same time again improve the quality of the product for its use as blast furnace burden.

The accompanying bibliography indicates the literature readily available to one interested in the subject. Unquestionably not only the industry at large but each individual plant involved, would receive much benefit if those most qualified were more free to honor the literature with their experience.

The author desires to express particular obligation to Mr. J. E. Greenawalt for the most painstaking instruction and helpful interest in the study of all branches of the subject.

Nearly all the references given in the bibliography have been studied either in their original or translated form; a few only have been seen in abstract.

Acknowledgment is accordingly freely made to each of the respective authors.

II.—Bibliography.

PATENTS.

1. U. S. Patent 600,347, March 8, 1898: Thomas Huntington and Ferdinand Heberlein. By mixing lime with lead ores, and, after a preliminary heating, forcing air through the hot mass, the ore is both desulphurized and sintered. The ore is first to be heated in a furnace and then transferred to another receptacle for the completion of the process.

2. U. S. Patent 705,904, July 29, 1902: Archibald Drummond Carmichael. By mixing gypsum with sulphide ores of lead he is able, after starting the reaction with heat, to prepare the ore for blast furnace work in one operation. "My invention consists, essentially, in dispensing with the preliminary roasting and in the complete desulfurization and cineration of the sulphides in the converter in one operation."

3. U. S. Patent 755,598, March 22, 1904: Adolf Savelsberg. It is found that by mixing limestone with the lead ores sulphides can be desulphurized in one operation. "The final product is a silicate consisting of lead oxide, lime, silicic acid and other constituents of the ore, which now contains but little or no sulphur and constitutes a coherent solid mass which when broken into pieces forms a material suitable to be smelted." He uses a layer of limestone, either above, or below, or mixed with the layer of fuel, to protect the grate.

4. U. S. Patent 786,814, April 11, 1905: Thomas Huntington and Ferdinand Heberlein. They improve their original process by quenching the preparatory roasted ore with water before finishing in the converter; find that it is not necessary to mix lime with the ore if the hot roast is quenched and combustion started anew in the converter. Their points of importance are pre-roasting, cooling, and then blowing in the converter.

5. U. S. Patent 808,361, Dec. 26, 1905: Herbert Haas. Patents a certain regulation of the blast or air supply as used in his sintering converter.

6. U. S. Patent 839,064, December 18, 1906: J. E. Greenawalt. Combines a porous hearth with drawing air and gas down through so as to collect and save values otherwise lost during roasting and ore-treatment.

7. U. S. Patent 839,065, Dec. 18, 1906: J. E. Greenawalt. In ore-treating processes the material is placed upon a porous bed and the products of roasting or ore treatment are drawn down through the bed. The illustrations and application relate to mechanically rabbled furnaces.

8. U. S. Patent 870,690, Nov. 12, 1907: Adolf Savelsberg. Patents a two-stage roasting with wetting of the charge between.

9. U. S. Patent 882,517, March 17, 1908: A. S. Dwight and R. L. Lloyd. In treating fine ores so as to roast and sinter them the surface from which the gases escape is restrained so that the whole mass is uniformly firm. The principles of sintering are elaborately expounded.

10. U. S. Patent 882,518, March 17, 1908: A. S. Dwight and R. L. Lloyd. The method of sintering with movable grate is devised. The grate carrying the bed of ore to be treated moves relative to the ignition, suction, feed and discharge; a continuous process.

11. U. S. Patent No. 916,903, March 13, 1909: F. D. Weeks. A blast-roasting machine.

12. U. S. Patents 916,391, 916,392, 916,393, 916,394, 916,395, March 23, 1909: A. S. Dwight or to A. S. Dwight and R. L. Lloyd. The three types of continuous machines, cylinder, annular table and straight line, are illustrated and described.

13. U. S. Patents 916,396 and 916,397, March 23, 1909: A. S. Dwight. In the first patent the ore is sintered between a grate and a restraining device covering the ore, for either continuous machines or intermittent units; in the second patent the upper surface is restrained by a layer of coarse ore.

14. U. S. Patent 942,052, Dec. 7, 1909: H. C. Bellinger. A blast-roasting furnace.

15. U. S. Patents 951,198 and 951,199, March 8, 1910: W. G. Perkins and M. L. Requa. A new machine and method for blast roasting.

BOOKS ANE PAPERS.

"Lead Smelting and Refining." By W. R. Ingalls. Published by Eng. and Min. Jour., 1906.

Part V, Lime-Roasting of Galena, pp. 113-211, has the following reprints following the introduction:

1. Introduction of the Huntington & Heberlein process into the United States by the American Smelting and Refining Company.

2. "Lime-Roasting of Galena." By W. R. Ingalls. Eng. and Min. Jour., 1905, LXXX, 402.

3. "The New Methods of Desulphurizing Galena." By W. Borchers. Eng. and Min. Jour., 1905, LXXX, 116. Original in Metallurgie, 1905, II, 1-6.

4. "The Lime-Roasting of Galena." By W. M. Hutchings. Eng. and Min. Jour., 1905, LXXX, 726.

5. "Theoretical Aspects of Lead-Ore Roasting." By C. Guillemain. Eng. and Min. Jour., 1906, LXXXI, 470. Original in Metallurgie, 1905, II, 433.

6. "Metallurgical Behavior of a Mixture of Lead Sulphide and Calcium Sulphate." By F. O. Doeltz. Eng. and Min. Jour., 1906, LXXXI, 175. Original in Metallurgie, 1905, II, 460.

7. "The Huntington-Heberlein Process." By D. Clark. Eng. and Min. Jour., 1904, LXXVIII, 630.

8. "The Huntington-Heberlein Process at Friedrichshütte." By A. Biernbaum. Eng. and Min. Jour., 1905, LXXX, 535.

9. "The Huntington-Heberlein Process from the Hygienic Standpoint." By A. Bierbaum, Eng. and Min. Jour., 1905, LXXX, 680. Both from Zeit. Berg. Hütt. Salinenwesen pr. St., 1905, LIII, 219.

10. "The Huntington-Heberlein Process." By Huntington and Heberlein. Eng. and Min. Jour., 1906, LXXXI, 1005. Original in Oest. Zeit. f. B. u. H., 1906, LIV, 631.

11. Editorial on the making of sulphuric acid at Broken Hill. Eng. and Min. Jour., 1904, LXXVIII, Aug. 11.

12. "The Bradford-Carmichael Process." By D. Clark. Eng. and Min. Jour., 1904, LXXVIII, 708.

13. "The Carmichael-Bradford Process." By W. R. Ingalls. Eng. and Min. Jour., 1905, LXXX, 778.

14. "The Savelsberg Process." By W. R. Ingalls. Eng. and Min. Jour., 1905, LXXX, 1067.

15. "The Lime-Roasting of Galena." Trans. Am. Inst. Min. Eng., 1907, XXXVII, 627.

The Mineral Industry under "Progress in the Metallurgy of Copper," by L. S. Austin, and under "Recent Improvements in Lead Smelting," by H. O. Hofman, since the development of blast-roasting, has given each year abstracts of the more important papers appearing in the literature. The references are very complete and doubly valuable because of the comments of the accomplished reviewers. Most of the following papers are referred to in this publication, but the list has been brought up to date and each of the following is worthy of individual study:

16. "The Savelsberg Process." By W. R. Ingalls. Eng. and Min. Jour., 1906, LXXXI, 1136. Short editorial with three illustrations.

17. "The Dwight and Lloyd Sintering Process." By Arthur S. Dwight. The Mineral Industry, 1907, XVI, 380.

18. "Lead Smelting in Utah." By R. B. Brinsmade. Mines and Minerals, 1907, XXVIII, 216. Abs. The Mineral Industry, 1907, XVI, 665. Describes blast-roasting at Murray and Midvale.

19. "Lead and Copper Smelting at Salt Lake, I and II." By W. R. Ingalls. Eng. and Min. Jour., 1907, LXXXIV, 527 and 575. Describes the plants at Murray, Midvale, and Garfield.

20. "The Dwight and Lloyd Sintering Process." By Arthur S. Dwight. Eng. and Min. Jour., 1908, LXXXV, 649.

21. "Treatment of Sulphide Ores by the Huntington-Heberlein Process." By L. S. Austin. Min. and Sci. Press, 1908, CXVI, 641. Gives cost of producing sinter in a 50-ton Huntington-Heberlein plant.

22. "Laboratory Experiments in Lime-Roasting a Galena Concentrate with Reference to the Savelsberg Process." By H. O. Hofman, R. P. Reynolds and A. E. Wells. Trans. Am. Inst. Min. Eng., 1908, XXXVIII, 126, also p. 935. A series of 17 tests blast-roasting galena with lime in charges of a little over 1 kilo each.

23. "The Production of a Converter-Matte from Copper-Concentrates by Pot-Roasting and Smelting." By George A. Packard. Trans. Am. Inst. Min. Eng., 1908, XXXVIII, 633.

24. "Lead Smelting and Refining at Trail, B. C." By A. J. McNab. The Mining World, 1909, Sept. 4. A very satisfactory account of the Huntington-Heberlein roasting is included.

25. In "Mines and Methods," 1909, I, 6, C. T. Rice gives an excellent description of the Midvale plant of the United States Smelting, Refining & Mining Company.

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28. "Pot Roasting of Copper Ores in Australia." By S. Radcliff. Mining Science, 1909, LIX, 67. Abs. Mineral Industry, 1909, XVIII, 485. Original in the Australian Mining Standard.

29. "The Mining Industry in Queensland." By G. W. Williams. Eng. and Min. Jour., 1909, LXXXVII, 603. Abs. Mineral Industry, 1909, XVIII, 486.

30. "Trail Smelter and Lead Refinery." By J. M. Turnbull. Mines and Minerals, 1910, September, p. 121. An acceptable article which includes much material already previously given by McNab. (See above.)

31. "Notes on the Roasting and Sintering of Ores." By F. M. Smith. The Mining World, 1910, Sept. 10, p. 459. A short account of mechanical roasting and the Dwight-Lloyd installation at East Helena.

32. "Agglomerating Fine Ores and Flue Dust." By Herbert Haas. Eng. and Min. Jour., 1910, XL, 814. A rather general smelting discussion and a proposed blast-roasting plant suggested by the one at Torreon, Mexico.

33. "The Metallurgy of Copper in 1910." By L. S. Austin. The Mineral Industry, 1910, XIX, 181. Describes the Greenawalt process for roasting and sintering copper concentrates and flue dust at the Modern plant near Denver.

34. "Recent Improvements in Lead Smelting." By H. O. Hofman. The Mineral Industry, 1910, XIX, 436. Abstracts of the important papers appearing during the year are given as well as comments on the process of blast-roasting, especially the success of the Dwight-Lloyd machines. Exception is well taken to some of the statements asserted by Paul Richter which were based on his laboratory experiments with roasting. "Beiträge zur Theorie des Huntington-Heberlein Prozesses und der ihm verwandten Verblaseverfahren," published by Robert Noske, Borna-Leipsiz, 1909.

35. "Down-Draf: Sintering Experiments." By A. W. Hahn. Eng. and Min. Jour., 1911, XLI, 1208.

36. "Midvale Plant of the United States Smelting Company." By Leroy A. Palmer. The Mining World, 1911, May 27. A good account of the roasting and sintering is included.

37. "Speisses and Their Beneficiation." By C. Guillemain. Eng. and Min. Jour., 1911, XLI, 858, from Metallurgie, VII, 19. An account of how speisses have been successfully blast-roasted to remove arsenic.

38. "Lead (The Metallurgy of)." By Henry F. Collins. Published by Charles Griffin & Company, Limited, London, 2nd Ed. 1910. Contains an excellent chapter on blast-roasting.

39. "Operations of the Doe Run Lead Company." By Albert H. Fay. Eng. and Min. Jour., 1910, LXXXIX, 610. Describes the operating of the Savelsberg converting plant at Herculaneum, Missouri.

40. "Recent Progress in Blast-Roasting." By H. O. Hofman. Trans. Am. Inst. Min. Eng., 1911, XLI, 739. See also discussion, p. 915, by James W. Neill. A fairly complete review, but blast-roasting at Garfield and Tintic was soon discarded and details at Murray and Midvale plants are not wholly accurate.

41. "The Sintering of Fine Iron-Bearing Materials." By James Gayley, Bulletin Am. Inst. Min. Eng., 1911, 56, 631. Records the success of the initial trying-out of the Dwight-Lloyd machine on iron flue dust and fine ore.

42. "Recent Progress in the Roasting of Sulphides." By Leon Guillet. Revue de Metallurgie, 1911, 8, T, August, pp. 561-605. An earlier and briefer paper entitled "Progresos de la Metalurgia del Plomo" by the same author appeared in the Revista Minera, Metalurgica y de Ingenieria, 1911, June 1, 8, 16, 24 and July 1, pp. 277, 292, 307, 318 and 330. Covers the developments in Europe, in particular the installation of the three annular Dwight-Lloyd machines at Bindsfeldhammer, near Stolberg, while some references are also made to American practice.

Note.—At the time of reading the proof, this February, we are able to add two recent papers on the subject: A. S. Dwight, in the *School of Mines Quarterly*, Nov., 1911, gives a good review of all roasting methods from the efficiency point of view; and an editorial correspondence in METALLURGICAL AND CHEMICAL ENGINEERING, Feb., 1912, gives a valuable account of the operation of the Dwight-Lloyd machines at Salida, Colo.

III.—The Chemistry of Blast-Roasting.

1. *The Elimination of Sulphur.*

According to locality and conditions the charge which it is desired to roast and sinter may consist of any of the intermediate mixtures between the limits of rich galenas on the one hand and nearly pure pyrite on the other. The general

experience, as gleaned from the literature, is that with rich galenas or high lead mixtures the charge, to be successfully blown, must be comparatively low in sulphur. The upper limit is given by some operators as 9 per cent, while with possibly somewhat lower leads other operators can blow mixtures with up to 12 per cent. With comparatively low lead in the mixture, the remaining sulphur being chiefly as pyrite or chalcopyrite, charges containing up to 20 per cent sulphur are successfully blown.

Evidently the two predominating factors as to the amount of sulphur admissible are, first, the fusibility of the charge, and, secondly, the form in which the sulphur exists. A charge which is as easily fusible as must necessarily be the case when the lead is present to the extent of 40 per cent or more, or which consists largely of leady matte, in both of which cases the sulphur to be removed must first be fully oxidized, must have the sulphur content already low for good results in a roasting pot. There is evidently no avoiding of this conclusion. In the installation of the three annular down-draft tables at Bindsfeldhammer, as described by Guillet in his paper, pre-roasting is first practised. With the pyritiferous mixtures, which are inherently more difficult to fuse, we also are favored with sublimation of the first easily discarded atom of sulphur in pyrite. This sulphur which is driven off by heat without supplying any heat by its oxidation is easily observable during the roasting of this class of material. With up-draft pots it is of little significance, but with down-draft sintering it is a source of much clogging of the pipes and of the fan if that be placed in close proximity to the unit.

The sublimation of sulphur is a condition to be met in the planning of down-draft construction, provided that pyrite may be a constituent of the charge, and in considering the heat evolution of the charge mixtures we may allow that a definite percentage of the sulphur will not be oxidized so as to supply its heat value to the raising of the temperature of the charge.

Experiments with 2-ton charges show that for the elimination of sulphur from leady copper matte by down-draft the best diluent is iron ore; silicious ore makes a somewhat more satisfactory sinter cake, but the sulphur in the product will be somewhat higher. Limestone makes a good diluent also, from the structural side, but the sulphur in the product is much higher than with silicious diluent. A matte containing lead, 22 per cent; copper, 15 per cent; and sulphur, 22 per cent, when diluted with an equal or large and varying amounts of limestone and a little silicious material, gave uniformly about 9.0 per cent sulphur in the product; silicious dilution alone gave good sinters with less than 7.0 per cent sulphur, while diluting with equal parts of silica and iron ore made a product, good in its physical properties, and containing only 4.0 per cent sulphur.

Repeated tests by Greenawalt have uniformly indicated that limestone is not a good diluent for pyritiferous charges. Other investigators have come to the same conclusion when using this kind of material.

The secondary reactions so prominent in the hand roasting of lead ores in reverberatory furnaces which furnish large quantities of sulphur trioxide are largely subordinated in up-draft roasting; with the still more expeditious results attained with down-draft work the production of sulphur trioxide is again diminished. The exact amount in the gases from down-draft units is not known to the author, although certain experiments indicate that it is very low, indeed. This is a most important consideration where the gases are to be diffused over cultivated areas or are to be neutralized and freed from the trioxide.

The volatilization of lead and silver values, once such a handicap to all roasting of lead ores in reverberatory furnaces, became of much less significance with the introduction of up-draft blast-roasting. Without a thought we charge into the pots ores for roasting which would undergo serious loss in a hand reverberatory; rich concentrates only will commonly be reserved for briquetting. With the advent of down-draft methods no reservation of ores on account of loss by volatilization will likely be required. Little data is as yet available, but there is every indication that the loss is quite insignificant. In ordinary pot or stall roasting the chief loss occurs because of the inevitable formation of blowholes and the overheating and burning which result to the charge about them. In down-draft work the formation of blowholes is automatically inhibited; at any point of too much heat evolution and consequent fusion the suction pulls down the melt and the air is choked off; nothing comes of it except delaying the operation of roasting.

Aside from this automatic inhibition of blowholes the charge is exposed to heat for a much shorter time than in the thicker charges using up-draft; instead of breaking out to a free surface the gases are filtered through the comparatively cool inert layers protecting the grate, some small portion of fines is made during the charging and discharging of the cake, but flue dust as such is not made to an appreciable extent in down-draft practice.

To obtain a prompt and satisfactory roast the required amount of air must have access to all portions of the ore mixture; that this may be possible the charge must be so constituted that air *can* circulate to all parts, and, even more important, the air must have access to all parts *until the required elimination of sulphur is accomplished.* On the one hand, the charge must not contain so little sulphur that combustion shall cease and the material becoming dry and powdery prevent the passage of air over a certain area, nor, on the other hand, dare the sulphur be so abundant that the charge shall become overheated and, falling into a pasty mass, effectually clog the area against further oxidation.

Independent observers have come to the same conclusion, that passing air *through* the mass of a charge to be roasted is many times more efficient than passing air *over* the same material, however well it may be stirred.

This is the chief factor in all blast-roasting; sulphur elimination demands the passing of air; for the easy passing of the air the charge must be so constituted that it will allow the air to pass both at the beginning of the operation and during the whole course of the operation.

It is not difficult to construct suitable grates for either up- or down-draft types of equipment, nor is it difficult to make the charge permeable to the influence of the draft. Originally the permeability of the charge is mainly a matter of its moisture content. Whether it is coarse or fine is no matter as long as the original moisture is appropriate for its physical condition. A charge of material, each particle of which is of ½ in. size, will pass air readily, whether dry or whether containing 2 per cent of moisture; a charge of slimy concentrates and flue dust may be almost impermeable, whether dry or whether muddy, yet there will be some definite moisture content, let us say 12 per cent moisture, at which, if the charge is first well agitated in a mixer and then put to sinter, the batch will hold its shape, be quite porous and allow the required amount of air to pass easily enough.

It has more than once been stated in the literature that batches were too fine for sintering; it is also recommended for charges too high in sulphur that considerably more water might be added to keep down the heat by its evaporation. The fallacy of either opinion may be quickly enough dispelled by a few, even small-scale, experiments.

2. *The Formation of a Sinter Cake.*

It so happens that those conditions which produce desulphurization by blast-roasting also leave the roasted product as a sintered mass. This has been characteristic of every effort and all methods or particular furnaces. The sintered product has been found remarkably desirable for lead blast furnace smelting; furnaces smelt rapidly and well using this kind of roast; it is so desirable that it is worth taking pains to provide a sinter cake firm and porous.

As has just been explained under Chemistry, the conditions

governing the passing of air account for the desulphurization: the same conditions apply to the formation of the sintered cake, where the ore remains unroasted the product will be fines, where the heat has been too intense the material has fused together still carrying its sulphur and now in a massive condition, extremely hard and tough.

Irony materials, such as iron ores, may replace the silica for desulphurization but not for the production of a firm sinter. When iron ore would be added to the blast furnace charge along with the roast, for the production of a suitable slag it might be considered advisable to let the roast carry the iron in whatever excess should be necessary for the final slag. The conditions evidently regulating this refer both to this production of a firm sinter and to the question of the fine crushing of the materials. It would certainly appear poor management to crush iron ore, thereby making the sinter friable when silicious fines are already to be had by simply screening the available silicious ore which would otherwise be fed direct to the furnaces.

FIG. 1.—MASSIVE SINTER CHARACTERISTIC OF THICK CAKES AND UP DRAFT.

FIG. 2.—OPEN SINTER CHARACTERISTIC OF RAPID WORK WITH SUCTION DRAFT.

For the ores high in lead, above 40 per cent, and the sulphur reduced to about 9 per cent it has been found that if the remainder of the batch consists of approximately equal parts of silica, lime and iron the sinter will be of good structure and work well in the blast furnace. For the ores lower in lead the mixture to be sintered may well contain up to 30 per cent silica, a little iron ore not mattering much but addition of limestone being avoided.

The two following analyses will certainly produce matty sinters:

Insol.	Fe	Pb	S	Zn
22.0	17.8	23.0	18.6	4.3
20.5	28.8	10.1	20.4	6.0

The sulphur is so high that local fusion will take place and although the sinter will be firm and tough it will be massive in parts and contain layers or patches of matte.

Two batches for direct converting made up of concentrates, silicious ore and a little iron ore or other rather inert diluent analyzing as follows will produce excellent and cleanly silicious sinters:

Insol.	Fe	Pb	S	Zn
26.0	24.0	17.0	16.1	4.7
23.2	19.3	13.8	17.2	5.3

Given the conditions necessary for sulphur elimination, and enough heat generated to momentarily fuse the constituents, if enough silica is present to form a web-like skeleton other care is hardly necessary.

In large and thick charges there is always a particular tendency to heat retention and melting together. The thin beds made with down-draft equipment are easily superior in this respect.

Thin beds can not easily accumulate as much heat and with the best work the sintered cake should replace the original charge in almost its exact volume.

The sinter will be firm, cellular and porous, not containing any portions heavily slagged together.

These cakes should not be crushed in the heavy breakers which are required for "pot-roast" cakes but rather in suitable toothed rolls.

3. *Igniting.*

Hot charges as come from pre-roasting, either in reverberatory or mechanical furnaces, are extremely porous provided that the ore has been stirred and is not badly caked. The particles evidently touch without packing, exactly as when a cold, fine ore is moistened and mixed. In one case the particles are sticky and lightly held together because of their temperature, in the other case because the amount of moisture is exactly right. In all cases stirring and mixing must be done so as to lighten instead of packing the ore; the hot or moist ore once patted firmly together will be impermeable enough to any ordinary effort to pass air.

Following the original Huntington-Heberlein process that type of pot-roasting still begins the operation in the converters by blowing air up through a comparatively thin bed of red-hot pre-roast laid on a covering of clinker which protects the grate. The ignition starting afresh in the hot roast mounts through it and into the cold and moistened pre-roast or moistened "direct converting" mixture and soon accomplishes the desulphurization and sintering of the whole mass.

Obviously the cost of ignition by this method is nil, where preroasting is practiced.

Where pre-roasting is not practised the ignition of the charge comes from some outside source as from a mixture of fine coal, coke and pyrites as at the United States plant at Midvale, Utah (see papers by Palmer and others). After the grate has received a protecting layer of inert material, such as limestone or cold roast, the fire is applied in the form of hot and burning coal and coke, a slight blast is admitted, the pyrite is spread on, the charge is dropped on and the blast is increased.

Igniting is certainly a considerable item of cost when this method is used. In general, if the charge mixture itself does not ignite readily the cost is heavily increased. It is a pretty

sure method when enough fuel is thus used; almost any amount of water may be driven off or too dry charges will be sintered whether they support combustion themselves, or not. Other methods would be far cheaper for doing these kinds of work. An inherent difficulty with this method as with Huntington-Heberlein igniting is that results cannot be fully known until the cake has to be dumped.

A charge for down-draft sintering, which is composed of pyritiferous lead concentrates, fine silicious ore and other desirable diluent, if sulphur, silica, moisture and mixing are properly done and the section applied, will take fire immediately upon the application of a combustible material or a flame. Oil, sprayed from nozzles under pressure and confined over the charge with a hood, is the most suitable igniting agent yet found. The lighter oils possibly work a little better but partly refined or even thin crude oils are satisfactory.

A 2-ton charge with a surface approximately 5 ft. by 8 ft. may be ignited with a flame of atomized oil, sprayed from nozzles with a pressure of 90 lb., in 15 seconds. One minute is a long time to allow for such a flame to start even a poor charge; if the surface will not ignite sooner than this the heat will so dry the top that longer heating is useless.

Especially at night it is a beautiful sight to watch this sort of igniting. A bit of waste lights the oil as it is turned on, the suction draws down the flame, it roars a few seconds, is turned off, the hood is rolled away, exposing the whole surface, a half minute before cold, damp and inert, now seething with an intense combustion that quickly eats its way into the cake, leaving the exterior again black and dead but now hard and firm. Lacking unusual imagination the operation needs be seen for full comprehension. In case any spots are not well started they are best remedied by waiting a few minutes and then raking a little hot crust from other parts over the raw areas.

IV.—The Types of Roasting Units.

All the types of roasting units have been described in the literature; detailed descriptions, dimensions and operating methods may be found by referring to the individual papers. We shall concern ourselves here only with the principles involved and the cost of operation.

1. *Huntington-Heberlein Pots.*

This method of roast-sintering has gained a remarkable standing the world around, but is evidently yielding to the down-draft methods as at present solely represented in operation by the Dwight-Lloyd machines.

The particular points strongly in favor of the H. & H. method are large tonnage, ability to handle rough and ready charges and its adaptability to all sorts of ores and conditions.

The weak points are volatilization, massive and high sulphur sinters, large cost of pre-roasting, handling and breaking, production of considerable fines and poor working conditions.

Unquestionably a small plant may be rushed to a great capacity; 8-ton units can do fairly good work and turn out 16 tons of charge each per 24 hours, day after day. The product may be badly slagged and matted, sulphur may be high and much fines produced, but still tonnage holds high and roast is always on hand for the furnaces. This is an important consideration in any successful plant.

On the other hand, these very conditions are the weakness of the method. Any process equally reliable and of the same operating cost that does not volatilize several per cent of the lead, that does not make large quantities of flue dust and fines, that does not make massive and matted roast or that requires less attention and affords better working conditions, will be in line to ultimately supplant the process.

Up-draft plants are bad to work in. The slightest derangement of parts and the customary excess of blast over draft create a condition not alone uncomfortable, but one which the better class of labor is not prone to long endure. This is the smallest consideration in any plant, but in a careless moment of abounding fresh air and ignorance of the future we dare assert that it is a point against the H. & H. process in particular and against all up-draft sintering in general.

Metallurgists have asserted that the free and unrestrained upper surface of the charge in an H. & H. pot is the source of the blowing over of the dust, the volatilization, the blowholes, and the slagging and matting. It is not to be denied that the top of the charge is opened by the force of the blast, and that the upper portion is constantly poked and dug into by the man in charge in his efforts to prevent the blowholes; certainly the top layers remain more or less unagglomerated to the last, and must be poured off for succeeding charges. In spite of all this what ground is there to suppose but that this is not other than a result of a condition which develops in the bottom of the charge and finally breaks out at the surface? Local fusion and inequalities in the mixture inevitably induce a concentration of the blast into channels which wend their way up to the surface. The ideal condition described by Professor Hofman is certainly far from being realized in those plants where the miniature volcanoes break out an hour or two after charging, and must needs be continually demolished from then until the cake is declared to be finished. If the moisture is right and the mixing is thorough the blast will start right: it is then merely a question of how soon the oxidation is gaining around the edges and in cones throughout the interior. Long observation impels me to believe that most of the roasting is according to the latter interpretation.

The large cakes made in the regulation converters are difficult to reduce to the proper size for smelting. While still hot the cakes are tough and sticky and bad to handle, their whole interior is commonly red-hot, much water is needed and labor wastes itself. The 24-in. by 36-in. crushers commonly supplied for the crushing are also too small for the chunks, which ought to drop in easily. Breaking by dropping or letting a weight fall on the cake makes much fines, and is tedious.

Pre-roasting widens the range of the method in that it can treat higher lead and higher sulphur ores for any stated tonnage. The costs per ton of output, however, are certainly higher than for direct converting, and which method is the more economical will then evidently depend on the abundance and character of the ore supply. Against this process we must also charge the necessity of maintaining a briquetting plant if that is kept on account of ores too valuable to trust in the pots.

Pre-roasting is by no means an operation so cheap that its cost may be neglected. I have yet to be convinced that it is expedient to use it except in exceptional cases where the ore supply is remarkably one-sided.

During 284 working days the mechanical pre-roasting furnaces of an H. & H. plant required 3330 tons of coal and produced in conjunction with the pots 24,023 tons of product; direct converting material was also made to the extent of 22,580 tons, but the igniting material came from the same furnaces. If we allow for ¼ of this direct product as having been pre-roasted and the cost of coal as $3.00 per ton the cost will be:

$$\frac{24{,}023 + (22{,}580 \div 8)}{3330 \times \$3.00} = \$0.372$$

If we reckon power at	$0.06 per ton
labor at	0.06 per ton
fuel at	0.37 per ton
incidentals at	0.02 per ton
	$0.51 per ton

will be the cost of the pre-roasting.

If this amount of original sulphide diluted with ¾ of its weight of silicious and other material, ½ of which is to be briquetted at an equal cost, were direct converted over a primer of the pre-roast, the difference in cost might be roughly estimated thus:

Pre-roasting and blowing 100 tons at $0.95	= $95.00
Direct converting 175 tons at $0.75	= 131.25

If this charge of $36.25 be divided by 2, since half of the material would be briquetted at the same cost, it leaves $18.13, which, reckoned against the 37.5 tons of fine material, is evidently a charge of 48c. per ton. This is a considerable tax against the ore, but it must be considered that it is for the benefit of the entire blast furnace charge, let us say 600 tons. It amounts to some 3c. a ton thus disposed of, and the question is, has it benefited the entire charge to that amount. The answer is that taking the briquettes and fines from the furnace burden has made far more than this small benefit. The increased rate of smelting alone will amount to more than that. We must also admit the long campaigns, the general good behavior and excellent reduction of furnaces running fast and free.

If we wish to estimate the cost of the fuel for pre-roasting as charged against the entire output of the roasting plant,

$(3330 \div 46{,}603) \times \$3.00 = \$0.21$ per ton of product.

The cost of labor in a plant of this sort may be given as approximately according to the following table:

Labor per 24 hours, 200-ton capacity plant:

No. of Men.	Duty.		Price.		Shifts.		Amount.
1/3	Foreman,	@	$3.00	x	3	=	$3.00
1	Hoistman,	@	2.35	x	2	=	4.70
1	Pot Boss,	@	2.50	x	2	=	5.00
1	Craneman,	@	2.50	x	2	=	5.00
1	Fireman,	@	2.10	x	3	=	6.30
2	Trammers,	@	2.50	x	2	=	10.00
2	Pokers,	@	2.10	x	2	=	8.40
1	Bed Mixer,	@	1.90	x	2	=	3.80
1	Charge Man,	@	2.10	x	2	=	4.20
5	Laborers,	@	1.90	x	2	=	19.00
						Total,	$69.40

$$\frac{60.40 \times 284}{46{,}603} = 0.423$$

Operating cost of department, per ton product:

Labor	$0.42
Fuel	0.21
Power (estimated)	0.10
Bedding	0.05
Supplies (estimated)	0.01
Repairs (estimated)	0.06
Total	$0.85

At 400 tons capacity the cost should be not far from $0.69 per ton.

2. *Savelsberg Converters.*

Details of this little used process are to be found as indicated in the list of papers:

Ingalls, *Eng. and Min. Jour.*, 1905, LXXX, 1067.

Fay, *Eng. and Min. Jour.*, 1910, LXXXIX, 610.

Hofman, *Trans.*, Am. Inst. Min. Eng., 1910, XLI, 745.

It appears that with the high lead in the mixture the addition of limestone has no particular influence to retain sulphur. It is to be noted that before blowing the sulphur must be diluted to a comparatively low figure. The amount of limestone added is not very large, and that even then the tonnage is not much, if any, exceeding that of the H. & H. process. From the accounts of the manner of feeding the mixture into the pots the item of the cost of labor will be fully normal. As well be discussed more at length further on, the success of any roast-sintering process will depend largely on the mechanical and other engineering which is incorporated as a part of the process. From the published descriptions of this process it has little along this line to recommend it. From the account given by Hofman if the sulphur in the original charge is reduced from 11.20 to 2.30, and if 10 per cent loss in weight be allowed, this is equivalent to an elimination of about 82 per cent. This is exceptionally good work, and it is to be regretted that further details of operation and cost are not available. The latest *Mineral Industry* reports that these converters are being replaced by Dwight-Lloyd machines.

3. *Bradford-Carmichael Converters.*

This process, although reported as no longer used, is interesting both from its chemical and engineering side. Sulphuric acid was made from the gases generated, which were unusually rich in sulphur dioxide.

The cost data as supplied by Ingalls is in particular to compare with the cost of the other methods:

Labor	$1.80 per 8 hours.
Gypsum	2.40 per 2240 lb.
Coal	8.40 per 2240 lb.

	Per Ton Ore.
0.25 ton gypsum	$0.60
Treating gypsum	0.48
Drying ore and gypsum	0.12
Converting	0.24
Breaking sinter	0.12
0.01 ton coal	0.08
	$1.64

4. *Converter Stalls at Midvale, Utah.*

Aside from the manner of igniting the blowing of these converter stalls is quite similar to the blowing of H. & H. pots. The remarks made about the principles involved in the volatilization, production of flue dust, creeping up of the burning zone, blowholes, massively slagged and matted product, etc., apply to one as to the other. Possibly the material used at Midvale may have, in general, been lower in lead than even the "direct converting" as used in the H. & H. process, and if the sulphur may at times have been lower they are probably favored with long periods when sulphurs can be considerably higher, due to this very fact of low lead.

On a basis of 270 tons a day produced, the following details are probably a close approximation of the labor requirements:

1 Shift.		Per Ton.
1 Bin man, at $1.80	$1.80	$0.020
1 Scale man, at $2.25	2.25	0.025
1 Motor man, at $2.00	2.00	0.022
1 Mixer man, at $2.25	2.25	0.025
1 Overhead tram man, at $1.80	1.80	0.020
1 Ram man, at $2.25	2.25	0.025
4 Poker men, at $2.00	8.00	0.089
2 Clay men, at $2.00	4.00	0.045
1 Crane man, at $2.75	2.75	0.031
1 Crane chaser, at $1.80	1.80	0.020
2 Fire tenders, at $2.00	4.00	0.044
2 Crusher tenders, at $1.80	3.60	0.040
1 Hoist man, at $2.25	2.25	0.025
1 Stall boss, at $2.50	2.50	0.028
½ Services of foreman, at $3.50	1.75	0.019
1/3x1/3x2 Hopper cleaners, at $1.80	0.40	0.004
	$43.40	$0.482

Three shifts will require 3x43.40, or $130.20.

For tonnages between 200 and 350 tons a day about the same number of men will be required to operate the plant; with the less tonnage a few men might be laid off, and with the higher tonnage a very few extra men might be required. We can then say that for a plant of this type the cost for labor will vary with the tonnage within these limits:

200 tons	65.1c.	280 tons	46.5c.
210 "	62.0c.	290 "	44.9c.
220 "	59.2c.	300 "	43.4c.
230 "	56.6c.	310 "	42.0c.
240 "	54.3c.	320 "	40.7c.
250 "	52.1c.	330 "	39.5c.
260 "	50.1c.	340 "	38.3c.
270 "	48.2c.	350 "	37.2c.

These various items may be segregated:

Supplying	$0.112
Operating	0.207
Crushing and loading	0.116
Supervising	0.047
	$0.482

Total operating cost:

Labor	$0.48
Fuel for igniting	0.11
Power (estimated)	0.09
Supplies (estimated)	0.01
Repairs (estimated)	0.09
	$0.78

Using the labor figure of 38c. for a 350-ton basis makes a total of $0.68 when producing that amount of product per 24 hours.

Evidently the figures as given apply to operating costs only. The cost of labor is by far the greatest single item. This item, in any plant, is one quite accurately determinable and little subject to bookkeeping manipulation. The item of fuel is also comparatively easy of estimation, but is much smaller in its amount. The item of power may vary somewhat with the manner of accounting, but it is believed that the figure as given is sufficiently close. Supplies form a small but distinct item. The item of repairs, or of repairs and maintenance, as might be called, is the most difficult of any given to estimate properly. Into this item enters the efficiency of the carpenters, mechanics, boilermakers, blacksmiths and electricians, all of whom are beyond the jurisdiction of the department, and may run up costs independently of the good efforts of the roaster foremen and the men directly under them. Here also enters the method of accounts. Is the department to take a certain proportion of the maintenance crews or actual hours in service? Certain repairs may even be long postponed for a low item, the men meanwhile enduring inconvenience and by their good will or from their needs keeping operations going.

It is to be noticed that these costs are for all costs whatsoever in the blast-roasting department. As it will cost practically nothing to get the product into the furnace storage bins the original unloading charge is put against the smelting department, and costs begin with getting the ore out of the bins. As the material must necessarily be sampled whether roasted or not this cost, with which might be included any trifling screening cost, is also charged outside of the department.

5. *Dwight-Lloyd Sintering Machines.*

These machines, which are becoming fairly numerous about the country, cannot be spoken of from personal knowledge. Their operating cost is commonly stated to be from 50c. to 75c. per ton of product. As to exactly what items this includes little information is given.

The principles of the operation are certainly all well taken, and the actual cost will evidently depend largely on the mechanical arrangements of supplying, mixing, feeding and disposing of the product.

In general it may be said that simple devices and methods are most favored by smelter operators. Lead smelteries in particular often try out good and standard equipment, afterwards reverting to cruder methods.

By referring to the items of cost, as given for roast-sintering at two large plants already mentioned, it will be seen that the actual cost of labor for operating is a comparatively small item, the cost of supplying and crushing and loading being greater. Operating cost is a strong point with the new machines, and if this evident extremely low cost is not otherwise compensated for they will inevitably gradually attain better standing. Repairs and maintenance are a cost item which will be found interesting as the machines begin to wear.

Besides being of continuous operation and large capacity, the draft is induced by suction from below and the charge lies in a thin cake. These are all strong points, and indicate a great future for the method.

(*To be concluded.*)

The Flare of Furnace Walls.

BY CARL HERING.

In an article on the "Thermal Insulation of Furnace Walls," published in the February issue of this journal, page 97, some curves were given for determining, in a very simple way, the loss in watts through furnace walls of different sizes, thicknesses, thermal resistivities and temperature drops, based on the rigidly correct formulas; they, therefore, take into account the flare, which for thick walls becomes important.

These curves were given for spherical and cubical furnaces as these shapes are the only two general types for each of which the results can be given by a single curve; moreover, they are the most rational shapes; for others not differing too greatly from these, the results would probably not be far wrong if one assumed for the calculations a spherical or cubical one of about the same inside surface as that of the particular one under consideration.

The calculation with the aid of these curves is partly graphical and partly arithmetical. The curves replace the tedious calculation of the geometric mean of the inside and outside surfaces (the square root of their product) which is the rigidly correct section to use for determining the insulation resistance, hence the loss of heat.

The purpose of the present article is to give the analytical formulas, their deduction and their modifications. It sometimes becomes necessary to use them beyond the range of the curves or when the calculation involves each of successive layers of different resistivities; even when the insulating material is the same, its resistivity may be appreciably different at different temperatures, as was shown in that article, hence different layers may have to be calculated separately. Moreover, it may be desirable to allow for the resistance of joints also, between different layers, and these may have greatly different areas in different parts of a thick wall. As the loss of heat through furnace walls is by no means small, it may often be desirable to endeavor to predetermine it as carefully as the conditions warrant.

In such more intricate cases it is far simpler to calculate the resistances of the various parts, add them together, and find the loss once for all from this total, instead of calculating the flow each time for each part in accordance with the older method.

Some slight complications arise in the use of such formulas due to the calculation of the geometric mean, and to the difference between the relative and the actual wall thickness. The complications due to the latter may be avoided by a clear and concise statement of the formulas, and those due to the former by not reducing them to what the mathematicians would call their "simplest" forms, as taught in text books, but by using instead incompletely reduced forms which, though quite as correct, are much simpler to use. What the mathematician considers the simplest form is not always the simplest one to use in practice. In the present case, by retaining a quantity which could have been eliminated and is therefore theoretically an unnecessary complication, the calculations become in fact much simpler.

Let R be the total thermal resistance from the inside to the outside, either of the whole furnace wall or of any one or more layers; it will simplify calculations to state this in thermal ohms,* and this is assumed here. Let r be the resistivity (specific resistance) also in thermal ohms. Let d be the actual

*A thermal ohm is that thermal resistance through which one watt will flow when the difference of temperature is one centigrade degree.

inside diameter, or the inside edge if a cube; D the actual outside one; L the actual wall thickness. Either inches or centimeters can be used provided they are used consistently throughout.

The rigid law says that the mean cross-section of a flaring body like that of a sphere or cube is the geometric mean. Hence if s and S are the inside and outside surfaces the correct mean will be the square root of $s \times S$.

Hence for a sphere, whose surfaces are πd^2 and πD^2,

$$R = \frac{rL}{\sqrt{sS}} = \frac{rL}{\sqrt{\pi d^2 \times \pi D^2}} = \frac{rL}{\pi dD} \qquad (1)$$

The quantity L could be eliminated because it is a function of the two diameters, namely, $2L = d + D$, but it simplifies the calculations to retain it.

Similarly for a cube

$$R = \frac{rL}{6dD} \qquad (2)$$

These are rigidly correct; they are based on the assumption that the resistivity r is the same throughout.

In many cases the calculations are still further simplified by using the curves mentioned above. In these the thickness is given *relatively* to the inside diameter or edge; let this relative thickness be l, then $l = L/d$ or $L = ld$. This quantity l is, of course, independent of whether inches or centimeters are used.

Substituting this in (1) for a sphere

$$R = \frac{rl}{\pi D} \qquad (3)$$

but as $D = d + 2ld$

$$R = \frac{r}{d} \times \frac{l}{\pi + 2\pi l} \qquad (4)$$

Or for cubes (2)

$$R = \frac{r}{6d} \times \frac{l}{1 + 2l} = \frac{r}{d} \times \frac{l}{6 + 12l} \qquad (5)$$

Hence the heat flow in watts, $W = T/R$, when centigrade degrees and thermal ohms are used, is for spheres

$$W = T \frac{d}{r} \times \pi (1/l + 2) \qquad (6)$$

and for cubes

$$W = T \frac{d}{r} \times (6/l + 12) \qquad (7)$$

The curves above referred to give the values of the last factor in each of these, which is the part that is the same for all sizes, resistivities, or temperatures; hence they give the watts when T, d and r are each unity. The curves were calculated from these two formulas.

Hence by multiplying the watts given by the curves for any particular value of l, by the actual inner diameter or edge, and dividing by the particular resistivity of the material in thermal ohms, gives the watts per degree; multiplying this by the temperature drop gives the total watts.

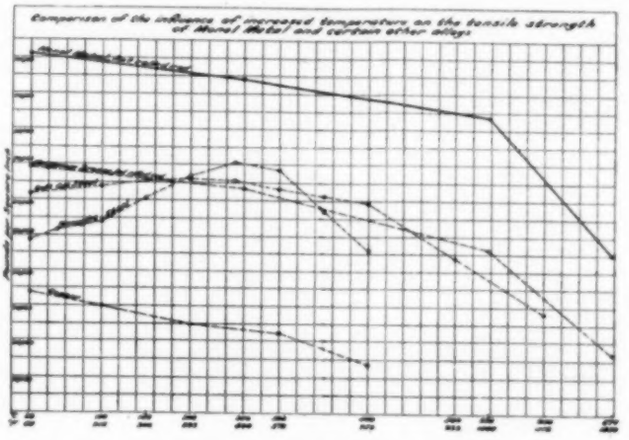
The number of "watts per degree," just referred to, is in fact the thermal conductance in thermal mhos* and is the reciprocal of the thermal resistance in thermal ohms.

Influence of Increased Temperature on the Strength of Certain Metals and Alloys.

The strength of various metals and alloys at ordinary temperature is well known and the selection of the proper-size piece of a suitable material is comparatively easy. Little is known, however, of the strength of these materials at higher temperatures and it is only by extended investigation of pieces tested while at these temperatures that a safe basis for proper selection may be obtained.

*Thermal Resistance and Conductance; the Thermal Ohm and Thermal Mho. Metallurgical & Chem. Eng., January 1911, p. 13.

A reference to the accompanying curve will show the relation that the strengths of certain alloys at ordinary temperature bear to their strengths at higher temperatures. These results show that Monel metal retains over 80 per cent of its elastic limit and tensile strength at 1000° Fahr., while at that temperature steel retains only 71 per cent, manganese bronze



TEMPERATURE AND TENSILE STRENGTH.

60 per cent, and copper has lost 60 per cent of its strength at 750° Fahr. This is accompanied in the Monel metal with no less, indeed with practically no change, in ductility.

The Solidification of Iron Castings.

In the course of a recent lecture before the London Branch of the British Foundrymen's Association Prof. T. Turner, of Birmingham, said it was well known that iron castings were smaller than the molds in which they were formed, and the shrinkage had to be allowed for in making the patterns. He used the term "solidification" as including all the changes which took place between the pouring and the removal from the mold. Shrinkage varied with the hardness of the iron and with the rapidity of chilling. Total shrinkage was ascertained by casting a standard bar between iron yokes and comparing the measurement of the distance between those yokes and the length of the cold bar. During the process of solidification in the mold, cast-iron underwent expansion at several stages, and the total shrinkage was the algebraic sum of the several expansions and the uniform contraction.

With an extensometer recording temperatures and expansion simultaneously it was shown that ordinary soft, phosphoric gray irons expended three times and that those three expansions all occurred after the bar had become solid from the ordinary point of view. The expansions reached their maximum degree respectively at about 1100° C., 900° C. and 700° C.—points which seemed to correspond with the separation of primary coarse graphite, the phosphorus eutetic, and pearlite.

The lecture was illustrated by diagrams and photo-micrographs showing the stages of crystallization of cast-iron and the nature of the three expansions, and explaining the difference between the effects of crystal thrust at the beginning of the solidification and that of the subsequent expansions.

Calorimeters.—An interesting bulletin has just been issued by the Schaeffer & Budenberg Mfg. Company, of Brooklyn, N. Y., on three new types of calorimeters. No special skill is required in their use on the part of the operator. The first is the "throttling calorimeter" for determining the amount of moisture in saturated or supersaturated steam, its operation being based on the measurement of heat. The second is the "separating calorimeter," which serves the same purpose, the percentage of water being determined, however, by mechanical separation of the water from the steam. The third is a "coal calorimeter" for determining directly the calorific value of coal.

New York Meeting of the American Institute of Mining Engineers.

THE 102nd meeting of the American Institute of Mining Engineers was held in New York City on February 19, 20, 21. The immense general interest taken in this meeting was evidenced by the excellent attendance from all parts of the country. All business and professional sessions and social functions were equally well attended. While the number of those who registered was 175, a fair estimate of the actual attendance is 250.

Business Meeting.

The meeting was a huge success in every sense and its keynote was harmony. Harmony prevailed at the business meeting which was held on the morning of Tuesday, February 19, when it had been thought that opposing views might clash on the framing of the future policy of the Institute, especially on the proposed amendments to the Constitution as to the reclassification of members with a raise of annual fees and the change of the name of the Institute to "American Institute of Mining and Metallurgy."

That the "insurgents" were in majority at the present meeting, however, became evident at the election of directors. There were two tickets in the field. That of the "insurgents" received 45 votes of those present and 881 proxies, that of the "regulars" 29 votes of those present and 310 proxies (these being the mean figures for the three candidates respectively). The directors elected are Messrs. E. B. Kirby, Geo. C. Stone, and Chas. F. Rand. On motion of two of the gentlemen on the other ticket the election (irrespective of proxies) was made unanimous.

For the election of officers and councillors, there was only one ticket in the field and the following gentlemen were elected: Prof. James F. Kemp, president; Messrs. Benj. B. Thayer, Karl Eilers, and Waldemar Lindgren, vice-presidents; Messrs. Jos. W. Richards, John H. Janeway, Jr., and Sidney J. Jennings, councillors; Dr. Joseph Struthers, secretary.

Dr. Ledoux, as chairman, expressed his gratification at the great interest that had been taken and at the spirit of harmony, of absolute good feeling and of community of purpose exhibited by all of the members.

PHOTOGRAPH TAKEN AT BANQUET OF AMERICAN INSTITUTE OF MINING ENGINEERS.

These controversial matters did not come up for discussion and decision, since a motion was proposed by Mr. J. Parke Channing, to have all the matters referred to a special committee of five, very carefully selected as to its membership so as to be representative of the whole Institute. The scope of the investigation to be carried out by this committee is very broad. It will report to the Directors of the Institute and an adjourned business meeting of the Institute will be held on June 3, 1912, or on such later date as may be mutually agreed upon by the committee and the directors. At that meeting the proposed amendments to the Constitution are to be finally considered. This committee consists of Messrs. Corning, Kemp, Ledoux, Wm. H. Nichols, Jr., and Stone.

This peaceful solution of the controversy which had been originally proposed by the "insurgents," found just as much favor with the "regulars" and indeed with everybody, so that Mr. Parke Channing's motion was unanimously carried.

Social Functions.

The convention was informally opened on Monday evening, February 19, by a social meeting and smoker of the members at the rooms of the Institute, which had been nicely decorated with palms and flowers. There was good feeling all around. An address of welcome was made by Mr. Benjamin B. Lawrence, as chairman of the local committee.

Mr. Charles Kirchhoff, the retiring president of the Institute, presented an elaborate address on the aims and ideals of the Institute, laying particular stress on the differentiation of the duties of the Instiute toward its several types of members—mining engineers, geologists, and metallurgists.

Prof. James F. Kemp followed with a very interesting and humorous talk on some of his adventures in the Canadian

Rockies and in Mexico. This address was illustrated by lantern slides, some of which were very amusing.

On the evening of Tuesday, February 20, a very enjoyable subscription dinner was held at the Hotel Plaza. It was well attended and was graced by the presence of many ladies. Mr. Charles Kirchhoff, as toastmaster, was excellent in the pointed brevity and quiet humor of his introductions.

The first speaker was Director George Otis Smith, of the U. S. Geological Survey, who spoke of the ideals and the career of the Survey.

Mr. W. L. Saunders gave in a most interesting speech some of the impressions which he got from his recent trip around the world. He emphasized that the industrial supremacy in the world is now passing over to Germany and to the United States and spoke in glorifying terms of the engineering capabilities of the German people, especially in mechanical engineering and engine building. The Germans are a practical people, they take nothing for granted; whenever they find anything they try to improve on it. As to the labor problem, cheap labor is not the foundation of success in manufacture. In Germany the price of labor has gone up rapidly in recent years in proportion with efficiency. It is efficient labor that really counts. Mr. Saunders concluded with some remarks on the tariff problem as the most important question now before the American people, but he thought that it would be a great mistake to take the tariff wall down all at once.

Dr. R. W. Raymond made an eloquent address, as only he can make, on the mining engineers' wives and the mission they faithfully and untiringly fulfill.

Colonel Alexander M. Hay, of Toronto, exhibited clever literary ability by discussing in a decidedly witty address in rhymes the problem of "latitude," with special reference to a famous controversy concerning the discovery of the North Pole. "There is no latitude for liars at the Pole."

The concluding speech was made by the Secretary of the Canadian Mining Institute, Mr. R. Mortimer Lamb, who discussed the work and career of this Institute.

* * *

Sintering of Fine Iron-Bearing Materials by the Dwight-Lloyd Process

The first professional session was opened by President Kirchhoff on Tuesday afternoon. It was chiefly devoted to a series of papers on agglomerating and briquetting fine materials.

The first paper dealt with the "sintering of fine iron-bearing materials by the Dwight-Lloyd process," the author being Mr. J. G. Klugh, of the E. & G. Brooke Iron Company, of Birdsboro, Pa., at which plant the Dwight-Lloyd process is in actual successful operation. In the author's absence the paper was presented by Dr. G. W. Maynard. (Mr. James Gayley's former paper on his application of the Dwight-Lloyd process to iron ores and concentrates was discussed and commented upon in our Vol. IX, page 431.)

According to the principle of the process, the finely divided iron-bearing material to be sintered is intimately mixed with the amount of carbon requisite to produce the sinter. This mixture is then moistened and deposited upon the machine in a uniform layer, the upper surface of which is ignited by a small flame of intense local heat, and the combustion of the intermixed carbon is effected in a progressive downward course by a current of air flowing in the same direction through the permeable mass. The energy from the combustion of each particle of carbon is expended and utilized directly upon the adjacent particles of iron-bearing material, agglomerating them into a coherent sinter.

In other words, the Dwight and Lloyd sintering machine is a piece of chemical apparatus in which to carry out the reaction between the oxygen of the continuous supply of air and the solid combustible material of the charge. The former is maintained in a zone of regular and continuous supply moving in the direction of gravity. The latter is, of course, intimately mixed with the material to be sintered. It logically follows that the capacity of the machine is limited by the weight of carbon burned per unit of time. It has been found that an amount of carbon equivalent to 8 per cent of the weight of the ore to be sintered furnishes ample heat for the production of the sinter. Carbon in the mixture in excess of 8 per cent performs the wasteful and deleterious functions of (1) consuming proportionately longer time for its combustion; (2) raising the temperature of the sinter produced to its fusing point, thus enveloping unconsumed carbon, and (3) raising the temperature of the grate bars and other parts of the machine. Besides which, the minimum consumption of carbon, consistent with maximum production and most desirable quality of product, is desirable for economical reasons.

From a given material, properly mixed with the correct percentage of fuel, the character of the sinter produced depends wholly upon the material itself, being unaffected by the speed of sintering or velocity of air supply; but its amount per unit of time depends upon the rate of combustion of the intermixed fuel, which, in turn, depends upon the rate at which the oxygen is brought into contact with fresh fuel, as the inert products of combustion are taken away by the action of the fan.

Thus the permeability of the bed of material, and the pressure drop between the atmosphere outside and that in the wind-box determine—other conditions remaining constant—the rate of the production of sinter.

Permeability is largely influenced by the moisture added. The function of the water is not entirely understood. Primarily it causes a temporary agglomeration of the particles of the materials, thus preventing them from passing through the grates. The voids in the bed are then larger, offering less resistance to the gas current. It must be borne in mind that the amount of added moisture is not a given percentage by weight, but the quantity required to produce a certain degree of plasticity. For instance, to a fine hematic there could be added 15 per cent by weight of water before the material would cohere properly, but with magnetite, 4 per cent of added water produces the equivalent degree of plasticity. The regulation of this phase of the process, which is doubtless very important, has as yet been determined by rule-of-thumb only. This is done very well by ordinary workmen after a small amount of practice, especially upon a given material.

The diagram, Fig. 1, shows the relative arrangement of the product and crude material in the active sintering zone. The rectangle represents a vertical cross-section through the pallets over the wind-box. The sinter and ore mixture are seen to be separated by an imaginary line beginning and ending at two

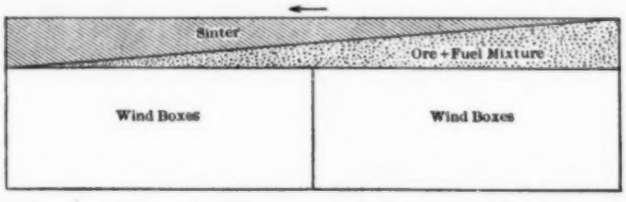


FIG. 1.—DIAGRAM SHOWING ARRANGEMENT OF MATERIAL IN THE ACTIVE ZONE, I. E., THE PALLETS OVER THE WIND BOXES.

respective zero points. This is about the state of all materials. When high-carbon materials are sintering the combustion of the carbon proceeds downward, progressively. Should high carbon follow low carbon directly, the only adjustment necessary is to slow down the speed of the pallets, allowing greater time for burning out the excess of carbon. The only condition under which residual carbon is left in the sinter is when the temperature of the sinter in the upper part of the bed is raised, so as to render it incipiently fluid, and permit it to envelop small amounts of carbon. This only occurs when the carbon is excessively high, say, about 20 per cent.

As to the desulphurization effected there have been practically no high-sulphur materials treated which were not brought down as low in sulphur as the best Lake ores. No special adjustment or preparation of the ores is necessary to effect desulphurization; the sulphur is eliminated simultaneously with the sintering. The desulphurization is not dependent upon the form or quality of the ore. Pyrites cinder has been brought down from 4.41 to 0.07 per cent, while magnetites have been lowered in sulphur from 3.50 to 0.15 per cent.

An analogy between the sinter made has been suggested by the Dwight and Lloyd process and mill cinder. No such comparison is practicable. The two materials are wholly different in origin and appearance. Whenever mill cinder or puddle cinder has a small amount of entrapped gases, producing pores, the walls of the pores are glazed and impervious to gases. In fact the material is a glass, which has been completely liquid. The Dwight and Lloyd sinter, made under normal conditions, has never approached the liquid state, and hence its particles are united by plastic coherence of their surfaces, and the cell walls have the microscopic porosity of easily reducible ores.

Microphotographs were given in the paper to show that there exists a positive distinction between the various forms of mill cinder and Dwight and Lloyd sinter. However porous the mill cinder may be, it can never approach in reducibility the product under consideration. Mill cinder is a satisfied chemical compound, only giving up its oxygen at high temperatures, and then affording a minimum area of contact for reaction of reducing gases. Dwight and Lloyd sinter is a physical agglomeration of material treated, being bonded of particles of original size.

Several thousand tons of sinter have been produced at the Birdsboro plant and used in the blast furnace. At all times the sinter has shown itself a superior and beneficial material of the blast-furnace charge. By reason of variations in the ore mixtures, no actual figures of the fuel saving effected through the use of the sinter are available; but several times as much as 10 per cent of easily reducible Lake ores has been replaced by sinter, and the management unhestitatingly declares that taking off the ore and putting on the sinter has the same influence on the fuel consumption as would result from taking off entirely that amount of ore burden.

At no time has there been any reaction in the furnace resembling that produced by mill cinder. When mill cinder is used with a burden of a high percentage of magnetic ores, a scouring action is often noted—especially when the mill cinder is put on to the amount of 12 per cent mass in one. There has never been at the Birdsboro furnace any scour action or descent of unreduced oxides since this sinter began to be used. According to all practised indications and theoretical considerations, there seems no doubt that a complete burden of Dwight and Lloyd sinter might be used advantageously.

The iron blast furnace stands on a high plane as regards metallurgical efficiency, but there are yet vast opportunities for improvement. Not the least of these lies in the due preparation of ores and fluxes prior to charging.

Mr. Gayley, in his Institute paper, quoted from Schinz's book on "The Action of the Blast Furnace" (1871) the following sentence: "A chemical reaction can only take place between two bodies, however great their affinity, if they are in intimate contact with each other; and the rapidity of this action will be so much greater the more numerous the points of contact are."

Mr. Gayley's purpose in this quotation was to lay stress upon the advantage of the Dwight and Lloyd sinter over other materials in giving a greater area of contact to the gases in the reducing zone. But the importance of this law is equal if not greater in its application to the reactions between solids in the fusion zone. And this is where blast-furnace practice seems farthest from the ideal at present. Materials are charged in groups of tons, which can only perform their proper functions by combinations between minute particles. It then seems only logical to say that, if greater intimacy of arrangement were provided between the bodies forming the slag, a distinct advantage would be gained, in that a slag of the desired chemical composition would be continuously formed in the zone of fusion. In present practice, there is an intermittent production —first, of a basic and then of an acid slag—all finding their way into the hearth, and making up a final fair average, but certainly not constituting, in the strict sense of the term, a continuous chemical operation.

The natural consequence of submitting to zones of varying temperature mixtures of varying melting points is a discontinuous and disproportionately wide zone of fusion. This source of irregularity is probably the source of many slips, scaffolds and other disturbances in furnace practice.

As a possible remedy for these conditions Mr. Klugh suggests a wider application of the Dwight and Lloyd sintering process. Let us assume, as a hypothetical case, a mixture containing all-fine ores of any specific burden (say, 40 per cent of the entire ore charge) and the flux of the entire burden (crushed to pass 10-mesh and under) to which the proper amount of carbon is added and sinter is produced. This sinter now contains: (1) The iron partly metallic and partly as either oxide; (2) the gangue and flux, not necessarily combined, but intimately associated and free from all CO_2. While the line is thoroughly calcined it differs radically from a separately calcined line, in that it is so agglomerated with other materials as to remain solid, instead of shaking and disintegrating.

The advantages to be expected from the use of such material in the blast furnace are:

A.—The solid mixture, containing uniform slag constituents, will carry them unseparated to the bosh, where their function will be performed continuously, instead of interruptedly (by "running ahead" and segregating).

B.—All the CO_2 having been eliminated, the solution of carbon by ascending CO_2 will be avoided.

Hence from there actions, $CaCO_3 = CaO + CO_2$; $CO_2 + C = 2\,CO$; there will be effected a saving of carbon equivalent to 12 per cent of the weight of the stone, assuming conditions of 1150 lb. limestone per ton of pig; 96 per cent of $CaCO_3$ in limestone; 85 per cent available carbon in coke, then:

1150 lb. limestone $= 1150 + 96 = 1104$ lb. $CaCO_3$.

$1104 + 0.12 = 132$ lb. carbon saved.

$132 \div 0.85 = 154$ lb. coke per ton pig saved.

C.—An increased reducibility of the iron so treated. (This has been conservatively estimated as 5 per cent better than the best available ore.)

D.—As a larger portion of the slag-forming constituents will have already been combined an amount of carbon representing difference between the heat for fusion and the heat of formation will be saved.

E.—Uniform delivery of properly porportioned materials to the different zones.

There has been set a numerical value for only the first case as it is almost universally conceded that the carbon solution factor is equal to the theoretical. Upon presenting this proposition to a well-versed student of blast-furance conditions he stated, "the furnace which does that will double its output." The author does not claim such sanguine advantages as this, but feels that an advantage would be gained far in excess of the expense of sintering the materials.

Materials otherwise regarded as unfit for the blast furnace may be readily and cheaply converted into a product most desirable for that purpose. For instance, as considerable flue dust carries an amount of carbon in excess of that required for sintering its own iron-bearing constituents, this may be used by mixing the flue dust with blue billy or magnetites, thus giving a product of uniform, cellular composition without cost for fuel.

In the extended discussion which followed Dr. Maynard gave the following figures of cost for the process. The first cost

of installation of one machine as installed at Birdsboro, with a capacity of 100 tons a day, is about $14,000, including superintendence, etc., up to finished erection; the cost of two such machines is estimated as $26,000. The cost of operation is 9 cents for labor charge, 3 cents for repairs, 24 cents for power, and 5 cents for ignition, or a total of 41 cents per ton.

Messrs. Richards, Schniewind, Eilers, Howe, Dwight, Chance, Vogel, Keith, and Cowles also participated in the discussion in which the chemistry of the sintering reaction and the reasons for the reduced fuel consumption were dealt with in some detail.

As to the production of fines in handling sinter produced by this process, Mr. Dwight stated that when properly conducted there should be no fines. There are less fines in the case of magnetite than in that of flue dust.

Very interesting samples of sinters produced in practice at Birdsboro by the Dwight-Lloyd process were exhibited.

Agglomeration of Fine Materials.

A paper by Prof. Walter S. Landis, of Lehigh University, gave a concise review of the general subject of agglomeration of fine materials, including both the molding and the sintering processes. In the absence of the author it was presented by Dr. J. W. Richards.

Fine materials have heretofore been wasted to a greater or smaller extent. A list of such fine materials includes:

1. Fuels, fines being produced in the mining and preparation and in the storing of coal and lignites.
2. Concentrates from ore-dressing operations.
3. Fine ores.
4. Flue dust. Prof. Landis estimates the flue dust made by the iron blast furnaces of this country in 1910 as 3,500,000 tons, carrying at least 35 per cent of iron. At a low value of 4 cents per unit of iron, this dust costs almost $50,000,000 for its iron content alone, besides the $1,500,000 worth of coke in it.
5. Scrap metal, such as filings, chips.

The processes of agglomeration of fine materials are divided by Prof. Landis into two groups.

In the first group certain properties in the material are utilized for producing the desired coherence without addition of any foreign binder. Thus ores carrying soluble salts, clay, easily hydrated compounds, etc., may be moistened with water and molded into form and dried. But while coherence is thus produced, the product has the great disadvantage that on heating to a temperature sufficiently high to drive off the water, the briquets fall to pieces.

Pressure greatly assists in developing the cohering power of many materials. Sometimes the best results are obtained with dry materials, sometimes a certain amount of moisture is necessary. The pressure so far used in such processes runs up to 2000 atmospheres.

Heating to a high enough temperature also brings materials into a pasty or semi-fused state so that they sinter into a more or less firm mass. When carried out in a rotary kiln, the process is called nodularizing. This same property may be taken advantage of by molding the material into briquets, using only a slight pressure to insure filling the mold, then burning the briquet so formed at a temperature that will insure cohesion. (Gröndal.)

Finally a combination of pressure and sintering may be employed.

In the second group which includes the greatest development of the briquetting process, a foreign substance is used as a binder. Some of the binders which have been employed are clay, lime, ground slags, natural and Portland cements, waterglass, kieselguhr, carnallite, tar, pitch, asphalt, petrolemum, sulphite residues, napthaline, parafine, molasses, resin, starch, and other materials.

The operations of this second group are divided into three sub-classes: (a) Processes in which the binder is mixed with fines and molded under low pressures; (b) processes in which higher pressures are used along with the binder; (c) processes in which a binder and pressure are used together, with a subsequent heat treatment.

Prof. Landis thinks not enough attention has always been paid to the selection of a binder for the particular case at hand. Too often is the intrinsic cost of the binder itself considered to be the main item in its selection, the subsequent cost it entails in passing through the smelting or refining furnace, being entirely overlooked. The waste-product possessing binding power may become very costly if it entails the production of an extra amount of slag to be smelted, or introduces in the furnace product a small amount of impurity which is difficult to remove afterwards.

Pressure increases the effectiveness of all binders, enabling one to obtain the desired results with the use of a minimum of extraneous material.

Finally the physical properties which a briquet should have are briefly summed up.

Sintering and Briquetting of Flue Dust.

A paper on the "sintering and briquetting of flue dust" was presented by Dr. Felix A. Vogel, of New York City. In the United States flue dust usually contains 20 per cent of coke and more than 40 per cent of iron. Estimating coke to be worth $3.25 per ton, and iron-ore 7 cents per unit, a ton of flue-dust, unless made available, represents a loss to the furnaceman of $3.50 per ton. This accounts for the first efforts to recharge the flue-dust into the furnace, either by moistening it down with an excess of water, or mixing it with clay to form balls or pulp, or treating it with limewater. These methods, however, have been practically discarded, as they failed to produce the desired economies.

The author enumerates the different conditions which must be fulfilled in an agglomerating process which is to be successful for the recovery of all the values in the blast furnace. Working along such lines American and European metallurgists have followed two different lines. In the United States it has sufficed to save the iron-content only, though not in the best possible form, while abroad the endeavor has been to make a high-class product for further use.

The author briefly reviews the different agglomerating and sintering processes which have been employed in this country, especially nodulizing in a rotary kiln, the Huntington-Heberlein process, the Gröndal briquetting process, the Dwight-Lloyd process, and the Greenavalt process.

Abroad the briquetting of flue dust has been more attractive than sintering. The Jime process mixes the flue-dust with from 5 to 10 per cent of hydrate of lime and, after briquetting the material is exposed to the air for a certain length of time, so that a carbonate is formed which is the binding medium. This binder will eventually act as flux and replace a certain amount of stone. The process, while having decided advantages, is quite cumbersome and costly, as the briquets must dry from two to four weeks under cover.

The pioneer process uses sulphide pitch (obtained from the sulphide pulp-mills) as a binder. It is an organic substance, rich in carbon and hydrocarbons, which will burn readily and thus increase the caloric value of the material. The flue-dust is pressed into briquets with from 4 to 8 per cent of sulphide pitch; they are quite hard and give fair results, but the process is expensive.

The Ronay process does not use a binder. The flue-dust is submitted to a very high pressure in a specially-constructed type of hydraulic press; the resulting briquets can be handled immediately and have proved very satisfactory. The process requires an expensive plant, however, increasing the operating cost.

The Schumacher process does not use what may properly be called a binder, but is based on the latent cementing actions existing in fresh flue-dust and which are made active by the presence of a small amount of a catalytic substance. Thus 0.25

per cent of magnesium chloride mixed with fresh flue-dust and from 6 to 10 per cent of water, pressed into briquets, will create a strong reaction, noticeable by the considerable heat developed; the briquets will be perfectly set and hard within a few hours. The process is very simple, an ordinary pug-mill being used in which to mix the material, which is subsequently pressed into briquets in a toggle-press and then loaded on cars to allow them to set.

Some flue-dust will react so strongly that a large amount of ore or coke-breeze may be added to the briquets; in these cases the flue-dust acts as a binder.

The Schumacher process requires an inexpensive plant and the cost of manufacturing is low. It is extensively used abroad and has replaced some of the other methods. The briquets of all-agglomerated materials have given most satisfactory results abroad and highest returns and values to the blast-furnace operators.

There were some very interesting exhibits of products of the Schumacher process.

In the discussion which followed Dr. Schniewind referred to a process used at the Krupp works in which granular blast furnace cinder high in lime is mixed with flue dust, subjected to superheated steam, and briquetted. Dr. Vogel stated that this process is no longer in use.

As to the use of chloride of magnesium, Dr. Vogel stated that it is used in Germany because it is a cheap by-product of chemical factories. In this country chloride of lime is used. As to the cost of the process, this was given as 40 to 65 cents per ton including royalty, in German and Belgian practice; in this country it is estimated as 20 to 30 cents. As to the nature of the catalytic actions, Dr. Vogel thought that three different chemical actions occurred, first cementing, second, oxidation of lower oxides, and third, hydration.

The Schumacher Briquetting Process.

A paper on "the Schumacher briquetting process" was presented by Dr. Jos. W. RICHARDS, of Lehigh University, who had studied the process in operation at the works of the Cambria Iron Company, in Johnstown, Pa.

Discovered in 1908, the process rests upon the observation of Dr. Schumacher that blast-furnace flue-dust, while possessing of itself no binding or cementing properties, acquires strong cementing properties if mixed with a very small amount of certain salt solutions, the amount of which is so small that the resulting cementation appears to be rather by catalytic action of the salts added than by the mass action of such chemicals. To be more specific, fresh blast-furnace flue-dust mixed with from 5 to 10 per cent of its weight of magnesium chloride or calcium chloride solution, that is to say with from 0.25 to 2 per cent of its weight of magnesium chloride or calcium chloride, acquires the property of setting within a short time, from 15 to 60 min., and forming a hard cemented briquet. Alkaline solutions appear to have no action such as described, and they even interfere, if present, with the action of the magnesium and calcium solutions.

When treated in this way the flue-dust briquet sets very hard, apparently from pure excess of cementing and setting-power. It is, therefore, possible to mix with the flue-dust a considerable proportion of inert ore or like material, which has no setting power by this process, and thus to make a compound briquet containing large quantities of ore mixed with flue-dust, but in which the flue-dust may be regarded as the cement or binding material. A particularly strongly cementing flue-dust will carry as much as three times its weight of fine ore or the like, and yet produce a satisfactorily strong briquet.

The Cambria Steel Company have installed at Johnstown, Pa., a small plant consisting of two presses, with a combined capacity of 250 tons of briquets per day. One of the presses is of the mechanical type, with hydraulic safety-regulator; the other is a Ronay hydraulic press. The first makes briquets, the size of an ordinary fire-brick, the latter makes briquets cylindrical in shape, 8 in. high and 8 in. in diameter, and using considerably higher pressures than the former.

As seen in operation early this month, the warm flue-dust from the dust-catchers was stored in one hopper, while fine Mahoning ore, containing from 10 to 12 per cent of moisture, and quite cold, was stored in another hopper. A rotating feeding device formed a stream of these materials, passing into the mixing-trough with ordinary spiral, into which at the same time was run a 30 per cent solution of calcium chloride, the flow of which was regulated by a hand valve. The trough fed the mixture directly into the hopper of the press. The flue-dust was warm, approximately at 90 deg. C.; the ore was ice cold; the solution was cold. The mixture fed to the press was just warm to the touch, the bricks going from the press were decidedly hot and steaming, and where loaded into the car they were quite hot to the touch and steamed vigorously.

This heating of the mixture as soon as compressed into the briquet is a characteristic of the process, and is an index of the rapid chemical reaction taking place which results in the cementing of the material.

The pressures used on the two presses seen were 550 lb. per square inch on the Brück.-Kretschle press and 1250 lb. per square inch on the Ronay press. The first press consumed 35 hp, with an output of 7 tons per hour; the second press 25 hp, with an output of 4 tons per hour.

The flue-dust carries from 18 to 20 per cent of coke, which is all carried into the mixture, said mixture being 70 of ore to 30 of flue-dust. There is from 5.5 to 6 per cent of coke in a finished briquet. The saving of this coke, 20 per cent in weight of the flue-dust used, represents sufficient value to pay for the entire cost of the briquetting of the dust. Undoubtedly the coke thus enclosed in the briquet acts on the ore in the furnaces, and reduces it with at least as great an efficiency as the "green" coke put into furnace. Therefore, if the furnace has sufficient smelting power, other things being equal, this coke of the briquet may be estimated as saving at least an equal weight of coke to the furnace.

As to physical properties, a briquet made of flue-dust, absorbed 11.5 per cent of its weight of water, representing 27 per cent of voids, a very satisfactory porosity.

Briquets one day old, hard dried, tested for compressive strength, gave a minimum of 445 lb. per square inch (31.8 kg. per sq. cm.).

Several hundred tons of these briquets were already in the ordinary furnace bins, ready for use in the furnaces, and appeared to have stood transportation to that point and dropping 10 to 20 ft., and also subsequent action of ice and snow for several weeks without deterioration; only a few were broken, and but little dust was made.

The operation of producing briquets in this manner, from flue-dust and fine ores, appears to have attained the maximum of simplicity, the operation consisting merely of mixing with solution and pressing into shape. Estimates of probable cost, if the operation were carried out upon a scale such as would suit a large blast-furnace plant, yielded as the outside figure 30 cents per ton, inclusive of depreciation, repairs, and interest on plant.

The paper elicited considerable discussion. Dr. Richards discussed the possible rationale of the process. It works on flue dust, but not on iron ore itself. Nor does it work on silica or alumina. But it works on chemically pure iron oxide. Pressure is necessary and the greater the pressure the harder the setting. It would seem that only at the points of contact the cementing action takes place. As a possible explanation of the catalytic action, Dr. Richards proposed a certain cyclic process based on the fact that calcium chloride hydrolizes.

Dr. Howe accepted this cyclic-process explanation as reasonable, but thought it could be somewhat simplified. He thought that all pure speculations on the best mechanical conditions of briquets carried no weight, but only experience on actual fuel consumption.

The question of the observed heating of the briquets and other points of the chemistry and thermochemistry of the process were the subject of some further comment by Mr. Dwight, Dr. Howe, Dr. Schniewind, Mr. Cowles, and Dr. Richards.

Gröndal Briquetting Process

A paper on the Gröndal briquetting process was then presented by Mr. Nils V. Hansell.

It was in 1899, when Dr. Gustav Gröndal was manager for the Pitkaranta Iron Works in Finland, that he built his first channel kiln for the briquetting of iron ore concentrates. Similar kilns had been used previously for the burning of clay bricks, but for the adaptation of the process to the treatment of iron ores and for the gradual development of a modified type of kiln suited to this special purpose, the honor belongs to Dr. Gröndal.

Since then a greater number of kilns have been built both in Europe and in this country. At present, there are sixteen plants in Sweden with together about 35 kilns and a briquetting capacity of about 400,000 tons a year. In Norway, there are three large plants—Sydvaranger with 8 kilns, Salangen with 4 kilns, and Dunderland with 4. In England 12 kilns have been built, all treating pyrites residues and blue billy. Italy has one plant, Spain has two, and in the United States there are six plants in operation or construction. In Canada the first one is being built by Moose Mountain, Ltd., for the treatment of magnetite concentrates.

The Gröndal furnace can briefly be described as a channel furnace, through which the ore, pressed into bricks, is transported on flat cars which form a continuous movable platform through the furnace. The furnace is heated by means of gas introduced through the raised arch at a distance from the entrance end of about two-thirds of the furnace length. The flat cars are usually built of structural steel with firebrick tops and are furnished on each side with flanges dipping into sand troughs. They are often built with overlapping ends so that when they are pushed against each other in the kiln, they separate completely, the upper part of the kiln, through which the gases pass, and the lower part, in which are the trucks and the wheels. The furnace walls are built double, with an outside wall of red brick and an inside one of fire brick; the two being separated by an air space, which assists in lowering the heat losses by radiation, and at the same time permits the walls to move independently of each other in expanding and contracting on account of changes in kiln temperature when the furnace is started and stopped.

FIG. 2.—INTERIOR VIEW OF A TWO KILN GRONDAL BRIQUETTING PLANT.

For obtaining the necessary temperature, almost any kind of fuel that has sufficient calorific value can be used. In Europe, the rule has been to use producer gas except at some places in Sweden, where the furnaces are built at iron works. There, a mixture of blast furnace gas and producer gas is burnt. Blast furnace gas alone is too diluted, so that the high temperature required for their action cannot be obtained unless it is enriched by some producer gas. When it is available, however, it is economical, as the coal consumption in the gas producer can thereby be cut almost in half. In this country, crude oil is used at two plants and natural gas at two. Local conditions govern the choice of fuel.

For different fuels and for different ores the construction of the kiln must be somewhat modified, although broadly the design is governed by the same principles. In the first third of the kiln the briquets are preheated by the escaping combustion gases; they pass through zones of increasing heat as the cars are pushed forward. In the combustion chamber the heat is maintained at about 2500 deg. F. When the briquets have passed the combustion chamber, they gradually pass through the cooling chambers, into which cooling air is blown by means of a fan. When they leave the furnace they are cooled off sufficiently to be unloaded directly into railroad cars. The arch over the cooling chamber is double and in some installations is made of corrugated cast-iron plates. Through the space between the double arches the air that is to be used directly for the combustion passes. On reaching the burners or the combustion chamber it has a temperature of 400 deg. to 600 deg. F. The outgoing combustion gases, when reaching the stack flue, generally have a temperature of 350 deg. to 500 deg. F. It can be seen that the furnace is built on the regenerative principle and that its heat efficiency is high. The walls, as have been said, are double, and the arch is usually covered with 8 in. or 10 in. of sand, so that radiation losses are low.

Mr. Hansell then discussed in some detail the chemical reactions which take place during the passage of the ore through it. As a rule, regardless of their original composition, the briquets are discharged as a peroxide of iron (Fe_2O_3). If the ingoing ore is a hematite, it is gradually dissociated to Fe_3O_4 as the ore passes toward the combustion chamber. In a nearly white condition, it is here met by a current of highly heated air. This gives the ideal conditions for a rapid conversion from Fe_3O_4 to Fe_2O_3. Here lies the secret of the Gröndal process. Briquets at a temperature of 2400 deg. or 2500 deg. F. are met by air of high temperature. The oxidation is almost instantaneous. Thereby an appreciable amount of heat is liberated, to the benefit of heat conditions in the combustion zone.

A quick oxidation of the Fe_3O_4 produces a blue crystalline Fe_2O_3. A slow oxidation will give a product that is earthly, amorphous and reddish in color. From this it is clear that it is necessary to keep a high temperature in the combustion zone. The color and condition of the discharged product tells whether the furnace is run properly. Dark blue, firm, and strong briquets indicate a right temperature; loose and reddish briquets a temperature too low.

When the ingoing material is a magnetite, it is probably oxidized to a certain degree on its way toward the combustion zone. This oxidation is probably followed by a dissociation as the briquets enter the hotter part of the furnace, but the Fe_3O_4 so formed is again oxidized to Fe_2O_3 before being cooled off and discharged. This last re-oxidation is so complete that the analyses usually show over 90 per cent Fe_2O_3 of the total iron oxide in the briquet.

It is probably these chemical reactions which account for

the agglomeration and the great strength of the briquets. In the re-oxidation of the lower oxides of iron, diminutive crystals of hematite are formed, growing out on the surface of each ore grain in the briquet. These small crystals interlace with each other or grow together, forming a strong bond between the grains. If the ore is not crushed fine enough or consists of rounded pebbles, the intergrowing effect may not be obtained, and consequently the resulting briquets will be loose and easily disintegrated. Such was the case with tests with the concentrated product from St. Lawrence magnetic sands. It was found necessary to pass a part of the iron sand through a grinding mill in order to get some fine material to fill the voids between the larger pebbles. With pyrites residues there has been the same experience. It is futile to try to briquet cinder from lump pyrites without first grinding it so as to pass, say, 20 mesh, if strong briquets are desired.

The briquets in passing through the furnace are desulphurized. At Bayonne cinder with 2 or 3 per cent sulphur is discharged with about 0.03 per cent.

A number of Gröndal briquets have been tested as to their suitability for blast-furnace and open-hearth use.

Last year a low-grade magnetite was crushed and concentrated in a large commercial test. A part of the concentrate was briquetted at Bayonne, N. J. Of the product about 7 tons were shipped in an open-bottom dumper car to Youngstown, Ohio. Here they were dropped in unloading from a trestle 30 ft. high onto an iron floor. A screening test of the unloaded material gave through a ½-in. screen 8.84 per cent of the total mass. These fines gave the following results at the sieve test:

On 8 mesh	69.70 per cent
On 20 mesh	16.66 per cent
On 40 mesh	4.55 per cent
On 60 mesh	0.81 per cent
On 80 mesh	0.80 per cent
On 100 mesh	0.60 per cent + 0.06 per cent
Through 100 mesh	6.66 per cent + 0.16 per cent
Total	99.78 per cent + 0.22 per cent

The porosity of the briquets, being the ratio of voids to the total volume of the briquet, is always over 20 per cent. This is a strong point in their favor, as it aids in making them easily reducible in blast-furnace operations.

It is the rapid reduction of the briquets in the blast furnace that has made them so popular in Sweden; 25 to 50 per cent of briquets in the burden of charcoal blast furnaces of that country has been demonstrated to reduce the charcoal consumption per ton of pig iron by 15 to 25 per cent. Fifty per cent of the briquets in the burden gives the best charcoal economy. In Sweden 500,000 to 600,000 tons of pig iron are produced yearly. As 300,000 to 350,000 tons of briquets are reported for the same time, containing in average 64 per cent iron, which corresponds to about 200,000 tons of pig iron, or about one-third of the total pig iron production of the country. This means a yearly saving for the country of 400,000 cubic meters of charcoal.

For a furnace in Belgium using imported Swedish concentrate briquets in a mixture with calcined Bilbao spathic ore the following figures were given: With 70 per cent of briquets and 30 per cent Bilbao ore in the burden, the coke consumption was reduced to 1760 lb. per ton of pig iron. With all Bilbao ore it had been 2398 lb. The iron content of the Bilbao was 50 per cent, of the briquets 65 per cent. The output of the furnace was increased 20 to 25 per cent, with a corresponding saving in wages and general charges.

Concerning one plant in this country it was stated that at Mayville the coke consumption in the blast furnace has recently been considerably reduced by their use. The briquetting plant there has been in operation so short a time that it is too early yet to expect authoritative figures.

For use in open-hearth furnaces, the briquets are eminently fitted, being rich in iron, free from noxious substances, in lump form, and of sufficient weight to sink through the slag cover.

Finally some practical details of the running of a briquetting plant were given, together with notes on improvements in detail which are being made with every new installation.

Fuel consumption is low. In plants treating magnetic concentrates the amount of coal used in the gas producers averages 7 per cent of the weight of the briquets, and there are plants in Sweden using as little as 5 per cent. At a plant in this country where pyrites residues are briquetted and crude oil is used as fuel, the oil consumption is stated to be 15 gal. per ton, which corresponds to approximately 10 per cent. It is obvious that if the ore is heavy, as magnetic concentrates are, the relative percentage of fuel used will be lower than when ore that contains a great amount of combined water and other volatile matter is treated. With such ore a shrinkage in weight of 15 to 20 per cent has to be taken into consideration.

Mr. Hansell had a fine exhibit of briquets made by the Gröndal process.

In the discussion which followed, Mr. Alfred H. Cowles said that the larger the furnace is made the more effective it is and the longer is its life. The enlargement in design represents progress made for this process in this country. Mr. Hansell stated that in Europe larger furnaces are also now being adopted.

Electrostatic Concentration.

A paper by Mr. HENRY A. WENTWORTH, of Boston, Mass., dealt with the electrostatic concentration or separation of ores. The subject has already been dealt with at some length in our Vol. III, pages 181, 1905 (Blake-Moscher process) and in Vol. VIII, pages 295 and 636, 1910 (Duff process), but Mr. Wentworth's paper contains considerable interesting new matter.

Electrostatic separation of ores in its present form is generally known as the "Huff process" from the name of Charles H. Huff, of Boston, Mass., through whose constant and persistent labors (with the invention of Clinton E. Dolbear as a basis) the successful commercial process embracing separative machinery and the various electrifying devices have been developed step by step, and the finances for the long period of development provided, and the method finally established and recognized throughout the world as an important and successful addition to the ore-dressing department of metallurgy.

The permanent field-success of electrostatic separation began in 1908 with a 20-ton Huff experimental mill built specially for the purpose by the American Zinc, Lead & Smelting Company, in Platteville, Wis., a plant which was a success from the start and was gradually increased as the market conditions warranted to 100 tons daily capacity of concentrates. Much credit is due to the management for their initial venture, and for their assistance in applying the process to field use.

Prior to 1908 electrostatic separators had been installed and operated (but for a comparatively short time, however) in a number of places; some under the patents of Mr. Dolbear by himself and associates, and some under the patents of Lucien I. Blake and Lawrence N. Morscher, by W. G. Swart, mining engineer, of Denver, who has always been a courageous advocate of electrostatic separation.

Due to the difficulties experienced in the generation of the electrical charges, the primitiveness of the separators, the wooden construction instead of iron as at present, the lack of control of the electrical fields and other difficulties overcome by the later inventions of the Huff Electrostatic Separator Company, electrostatic separation did not gain favor during those early field endeavors.

Without going again into details of the principle of electrostatic separation, it may simply be recalled here that it depends essentially on two principles.

First, there is the fact that two bodies charged with the same kind of electricity (either both with negative charges or both with positive charges) repel each other, while two bodies

charged with opposite electric charges (one with a positive and the other with a negative charge) attract each other. This is illustrated by the well-known old experiment in physics where an electrified rod is brought close to a suspended pith ball. The pith ball is first attracted, clings for a moment to the rod, and is then vigorously repelled. As the rubber rod approached the pith ball a charge of opposite kind, so called, was induced on the side of the pith ball nearest to the charged rod, and as unlike charges of electricity attract one another and as the pith ball was very light, it moved to the rubber rod. But pith, though not a good conductor of electricity, does, because of the moisture contained, conduct electricity appreciably, and it soon became as a whole charged similarly to the rubber rod, and away it flew.

The second fact of fundamental importance is that different materials have different electric conductivities, with

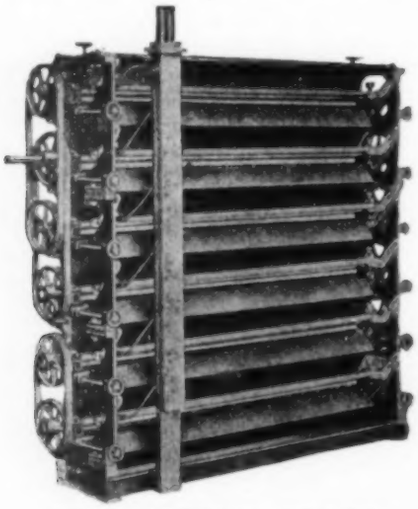


FIG. 3.—ELECTROSTATIC SEPARATOR.

the result that they require different lengths of time to be charged electrically. Assume that a mechanical mixture consisting of readily conducting and poorly conducting minerals (such, for example, as grains of copper and grains of sand) is dropped onto a metallic plate charged to a high potential. Immediately upon contact with the plate the better-conducting minerals become charged to the potential of the plate and are thrown vigorously from it. The poorer conductors require a much greater time to reach the electrical condition of the plate and therefore, if they are not given time to reach this condition, due to being removed from the plate, a separation is obtained. This method was utilized in the first simplest form of electrostatic separator.

FIG. 4.—ELECTROSTATIC SEPARATOR.

Two types of electrostatic separators are now in common use. A type consisting of six separating fields and a feed is illustrated in Fig. 3, which does not, however, show the large feed-hopper and the heavy base. The entire machine, with the exception of the attracting rods and their supports, made of metal, is electrically grounded, so that all portions which come in contact with the ore are metal and all parts which the operator is likely to come in contact with are grounded so that the operator gets no electric shocks in touching the machine. As it is desired to pass a thick sheet of ore through the separators, there is more or less interference of particles at each separating roll; hence it has been found advisable to use several rolls in succession, each contributing a part toward the complete separation. This is especially necessary in those cases in which it is desired to keep the ore coarse at the beginning, and thus have many attached particles to be re-ground. In order to have all the metallics come in contact with the separating surface a number of contacts at different angles is necessary. In this machine the only moving parts are seven steel rolls revolving in babbitted and greased bearings, so that the power necessary to drive the separator itself is extremely small.

Fig. 4 illustrates a diagrammatic sketch of a second form of separator which is finding particular favor in the separation of washed zinc-iron middlings below 20 mesh, which flow very readily. The advantage of this type is that there are no moving parts whatever, except the feed roll, and therefore there is a minimum of wear and required power. Not all material, however, will flow sufficiently freely without external assistance to be applicable to this type of separator.

The limits of size of particles which can be successfully treated by the present forms of Huff machines is from 6 or 8 mesh down to the limits of granularity of the material. Those slimes which are so cohesive that they do not move uniformly over inclined chutes cannot as yet be successfully handled. It is possible that this can be overcome by a special design of feed or separator. In Utah a table middling is being treated wherefrom the impalpably fine material has been removed during wet-table treatment.

The action produced in the usual type of electrostatic separator consists in electrically giving the relatively conductive particles a horizontal component of motion in addition to the motion produced by gravity. The less conductive particles not being thus affected are acted upon by gravity alone. The heavier the conductive particle, the stronger must this repellant force act to cause it to fall on the outer side of the divider of the machine. The repellant force is dependent upon the intensity of the electrostatic field. As electrostatic separation treats particles varying greatly in size, therefore, with a field sufficiently strong to repel the very heavy particles, there is danger of the finer, less conductive particles being thrown over also. Hence it is advisable not to have a feed in which the particles are too widely diversant in size, and therefore in weight. In practice the feed is screened into a few sizes, of which screening the following is an example: from 8 to 12; from 12 to 20; from 20 to 50, and through 50-mesh. When the difference in conductivity of the minerals is small, it is sometimes advisable to size somewhat more closely.

Ores are often found which crushing to 10 mesh, for example, will expose practically all the mineral, but will free but little of it. Because of the numerous separation fields to which the material is subjected while passing through the separator, all sides of every particle are brought into contact with the charging surface, and nearly every particle which contains an appreciable portion of exposed mineral will be repelled, while practically all the barren rock will be left in this coarse condition. The mineral can all be thrown in a concentrate, or the better part into the concentrate from the first two or three rolls, and the balance into a middling for recrushing.

The real difficulty in the introduction of electrostatic separation was the design of electrostatic separators which would not be influenced by the weather as frictional machines are. The complete solution of this important and difficult problem is the Huff machine, which is an electromagnetic generator, not a frictional machine. These generators work as independently of atmospheric conditions as does the ordinary lighting dynamo, and are capable of supplying any line or machine losses which may occur and yet hold the potential absolutely the same from day to day. Contrary to general belief, the difference of potential which is used in electrostatic separation is only from 18,000 to 25,000 volts, instead of the 100,000 volts or so earlier supposed to be necessary. There has been developed a small, compact 3-hp electrical set which is sufficiently large to care for the requirements of a 100-ton mill, and probably a much larger plant. Protective devices are placed between

the generators and the separators to prevent injury to workmen in case of accident at the separators and accidental contact with the high-potential line. In the four years of its field operation no one has been injured from this source.

As the effectiveness of any electrostatic separation depends on the differences in conductivity of the particles, it follows that there must be no extraneous factor interfering to affect the conductivity. Water is in the sense of electrostatic separation a good conductor. Therefore the particles must be dry. Some minerals dry very readily and remain so. Some are hygroscopic, and if allowed to stand cold for some time in a moist atmosphere, tend to collect on their surfaces an extremely thin film of moisture. However, experience has shown that by properly constructing the mill equipment so that the ore from the dryer passes immediately through its various paths, and is not permitted to lie for some time cold and exposed to the atmosphere of a room, very few minerals, and none of the common ones, offer any difficulty from this standpoint.

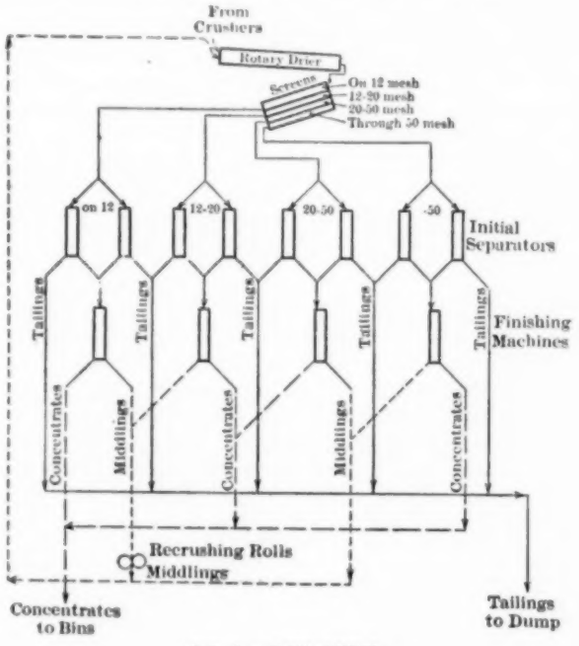


FIG. 5.—FLOW SHEET.

There seems to be a very erroneous idea of the cost of drying ore. Crude mine ore will run from 1 per cent to 5 per cent of moisture; 3 per cent being a fair average. This means 60 lb. of water to be evaporated from each ton of ore. A fairly efficient dryer will expel from ore carrying 3 per cent moisture 5 lb. of water per pound of coal, or, say 12 lb. of coal to dry the ore. With coal at $5 per ton this means a fuel cost of $0.03 per ton. Fine middlings from water-tables will sometimes carry as high as 15 per cent of moisture, and the fuel cost is therefore increased under those circumstances to 10 cents or 11 cents per ton with the $5 fuel cost, as with the higher per cent moisture the evaporating power goes up to 8 lb. of water per pound of ore.

For electrostatic separation all the minerals may be divided quite closely into two classes: first, those which are very easily charged and repelled, and, second, the very poor conductors which act as non-conductors. A table of this kind was already given on page 296 of our Vol. VIII.

Most of the copper sulphide minerals are excellent conductors, while most gangue rocks are poor conductors. Therefore, electrostatic separation is applicable to the general concentration of the sulphide ores of copper. It is particularly well-adapted to the concentration of copper minerals from the heavier gangue minerals, such as garnet, epidote, barite, the heavy basic rocks, etc. Large bodies of ores of this character exist, often as altered contact-deposits, though, except in cases where the grade is sufficiently high for direct smelting, they have been developed but comparatively little, because difficulty has heretofore been found in concentrating them.

Often a combination of methods is more efficient than a single process, the governing factor being whether the size of operations warrants complexity of procedure. Very large copper mills employing gravity concentration produce a concentrate which has been carried to the economical limit of purification by the method employed, but which is yet capable of mechanical improvement if a different property of the ore than its specific gravity be worked on. Electrostatic separation offers a means of reducing the silica and other objectionable constituents of the gravity-mill concentrates.

Zinc blende usually occurs in mechanical association with pyrite or marcasite, chalcopyrite, galena and one or more gangue materials. The specific gravity of these minerals is approximately as follows: Galena, 7.5; pyrite, 5.0; marcasite, 4.8; chalcopyrite, 4.2; blende, 4; ordinary gangues, 2.7. From the smelters' standpoint, it is essential (for maximum recovery of the metal at minimum cost) that the material going into the lead smelter be as free from zinc as possible; that into the copper smelter free from zinc and lead, and that into the zinc smelter be as high in percentage of zinc as possible. Of the latter, theoretically 67 per cent is the maximum, but only in exceptional cases (as in the Joplin and Wisconsin districts) is 60 per cent zinc product obtained commercially by any method, while 50 per cent zinc is for most complex ores considered excellent and 45 per cent good.

With ores in which the various mineral ingredients dissociate at a reasonable (30 mesh or coarser) degree of crushing, gravity separators (jigs or water-tables) will usually effect a reasonably efficient separation of minerals which differ in specific gravity by 1.5 points, but as the difference in specific gravity becomes smaller the effectiveness of separation in the above manner becomes less, so that with a difference of but one point, the effectiveness of the separation is poor.

By putting such a complex ore, suitably crushed, through jigs and over reciprocating tables the galena (with a portion of the pyrite and marcasite) is efficiently concentrated from the rest of the minerals. Likewise the elimination of the gangue rocks is also reasonably well effected. There are left together a greater portion of the pyrite or marcasite, the blende and the chalcopyrite (with, of course, a little gangue and a small amount of galena). Fortunately the galena (which is usually so small in amount as to be negligible except for the associated silver which often occurs in it), the pyrite or marcasite, and the chalcopyrite are excellent conductors of electricity, while the blende and the small amount of gangue rock are very much poorer conductors. In the utilization of this difference of conductivity is seen the application of the electrostatic process to this problem. This mixture after drying (but not roasting) is passed through the Huff machines and two products made, one for the zinc smelter and one for the copper smelter.

Blende often occurs in association with certain heavy gangue rocks whose specific gravity is so similar to that of the blende that separation by any gravity means is very ineffective. A method has been developed by the Huff company whereby the surface of the blende can be made conducting, after which the blende can then be separated electrostatically from the associated rock minerals. Two important instances of the above are blende and barite, and blende and fluospar.

In addition to the concentration or separation of the sulphides mentioned heretofore, the Huff process has proved that it also accomplishes the following results very satisfactorily: concentration of gold and silver pyritic ores, of antimony, arsenic, and molybdenum sulphides; concentration of graphite, of pyrite for sulphuric acid manufacture, hematite, manganese ores; the separation of galena and barite; purification of abrasives, natural and artificial; and the solution of many problems relating to the more rare minerals. Copper oxides, carbonates

and silicates can sometimes be concentrated, by first roasting to the conductive oxide, or reducing to the metal.

The process is essentially a dry one, obviating the troubles due to drought or freezing. The finished products require no further drying treatment prior to shipment or smelting to decrease freight expenses or fuel costs. Loss in slimes is avoided as the dust may be collected and is available for use if desired. There are no shaking or vibrating parts to the machines with the attendant wear and repair on the mechanism. The attendance necessary, after primary adjustment, is small and ordinary

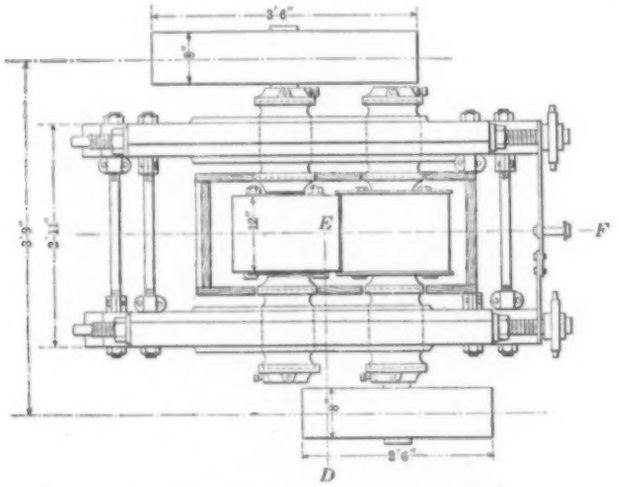


FIG. 6.—PLAN BELOW ROLL HOUSING OF FRAZEE CRUSHING ROLLS.

mill labor is suitable. The machines are readily sectionalized for light transportation and easily assembled. There are no complicated or intricate parts to get out of order and the separators are readily adjustable while operating.

In conclusion Mr. Wentworth pointed out that while drying is a requisite of electrostatic concentration, drying should not be confused with roasting. Not only is drying much cheaper, but roasting would have the great practical disadvantage that the particles would be liable to agglomerate and frit together and thus become unsuitable for electrostatic separation.

In the discussion which followed Prof. R. H. Richards remarked that electrostatic concentration, electromagnetic concentration, and flotation processes are a group of processes now coming to the front which do not depend on differences in specific gravity.

Roll Crushing.

The first paper of the Wednesday morning session dealt with "progress in roll-crushing," the author being Mr. C. Q. Payne.

Marked progress has been made within recent years in the field of coarse crushing by means of rolls, in response to the greater demands of modern mill practice, and this progress has been largely brought about by increasing the dimensions of the rolls and adopting a more massive construction, as well as a better design, combined with a wider choice of steel adapted to different ore requirements than has heretofore been available.

On the other hand, up to the present time rolls designed for fine crushing, where the ore requirements demand a maximum granulating effect and a minimum pulverizing or sliming effect upon the crushed product, have made little progress compared with rolls designed for coarse crushing. Mr. Payne points out that further advance in the art of fine granulation by means of rolls can only be expected by means of certain refinements of function whereby the roll-faces can be maintained parallel while they undergo wear.

These difficulties have been completely overcome, in Mr. Payne's opinion, in a recent design of rolls by Mr. J. S. Frazee, of Brooklyn, N. Y.

Figs. 6 and 7 show a pair of 18-in. x 12-in. rolls in plan and part sectional side elevation. The bearings of the roll shafts are supported in side frames. These are of cast iron, without tension rods, and are held together by stay-bolts and lock-nuts to permit the width between the frames to be slightly adjusted, if necessary, when the rolls are in place.

This construction has certain advantages for fine crushing over the single bed-plate or frame. It is lighter, less expensive, and permits the rolls to be completely boxed in by means of a wooden housing. This is hoppered at the bottom, and is connected with an exhaust-air pipe, through which the fine dust is drawn by means of a fan, thus keeping the mill space free from dust.

The bearings are of the solid type, lined with babbitt, and each provided with dust caps at each end. All four are movable in the guides provided for them in the frames. Two of the bearings are held against a spring pressure, and the other two against screw adjusting bolts, for the purpose of regulating and controlling the space between the roll faces.

These adjusting bolts are provided with sprocket wheels and an endless link chain, which is kept taut by means of a small idler between them. By inserting a long-handled spanner in openings in either of the sprocket wheels the adjusting screws can be moved in exact unison in varying the position of the roll space. This feature has been found a very convenient one, as it permits close and accurate adjustments of

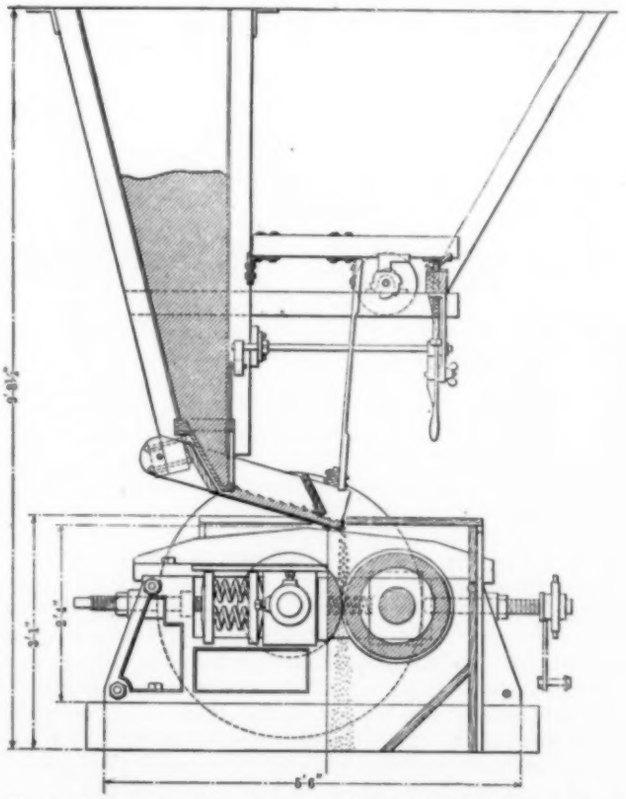


FIG. 7.—PART SECTIONAL SIDE ELEVATION ALONG LINES D, E, F OF FIG. 6.

the gap between the roll faces to be made while the rolls are running.

One of the essential features of Mr. Frazee's design consists, as shown in the illustrations, in making one roll-shell longer than the other, so as to permit flanges to form on it between which the shorter roll revolves with a slight clearance. The rolls are fed from a hopper, which is supported independently of the roll frame and in such a position as not to interfere with the removal of the rolls.

The lower end of the hopper is made of cast iron and is provided with a gate, which is operated by means of a shaft

and levers so that it can be controlled from the forward end of the roll frame. A feed tray is placed below the hopper so as to convey the material to be crushed from the hopper gate to the space between the rolls. This feed tray is supported at the back of the hopper by a pivot-bolt, which allows it to rotate slightly, and at the front end by means of a wooden connecting rod to a horizontal arm flexibly connected to a support at the front of the hopper.

When the front end of the arm is raised by the teeth of the revolving cam below it, the discharge end of the feed tray is caused to rise and fall with a jarring or bumping motion. This causes the material in the feed tray to be carried forward and to be discharged in a steady stream over the angle-iron which forms a lip or dam at the extreme edge of the tray. The bottom of the feed tray is made of sheet steel to permit the material to slide freely.

The amount of motion imparted to the end of the feed tray can be varied by means of an adjusting screw bolt, which regulates the height of the drop of the horizontal arm, and thereby also the effect of the bumping action upon the feed. A small lever at one side of the adjusting bolt permits the horizontal arm to be raised above the level of the cam, and in this way the feed can be instantly stopped and started.

It has been a common experience that as rolls are usually fed, even when great care is taken to secure a feed sheet of uniform thickness, the roll-shells will groove and wear more rapidly at their centers than at their ends. This is frequently caused by a difference in the rate of flow of the feed stream owing to friction of the sides of the chute or spout leading from the hopper to the rolls.

Another cause of greater wear of the central portion of the roll-shells is the greater mobility of the feed stream in certain directions than in others when subjected to a crushing pressure. This difference in mobility causes a difference in the amount of abrasion, and it shows itself by the cupping, grooving, or corrugating of the roll-faces along the center and by ridging at the edges of the roll-shell.

Mr. Frazee has found that the difficulty of irregular wear of the roll-shells can be entirely overcome by a close control over the sectional shape and area of the feed stream and by feeding a greater amount of material along the sides of the roll faces than at their centers.

The projecting side of the angle iron which forms the lip of the feed tray is cut down at its two ends so as to slope gradually towards the ends, leaving its full height only at the center. In this way the feed stream passing over the lip or dam is made thicker at the ends than at its center. Furthermore, to regulate the flow and upper contour of the feed stream, as well as the lower, a guide plate, which is usually hinged to the sides of the feed tray, is allowed to rest upon the surface of the feed-stream. This is provided with a wearing plate at its lower edge, and is so beveled at its two ends as to permit a somewhat greater freedom of flow of the feed-stream at its edges than at its center.

The cross-section of the feed-stream is adjusted to secure an equal abrading effect across the entire width of the roll faces, and thus to maintain them in perfect parallelism while they undergo wear. The exact cross-section of the feed-stream will vary somewhat with the hardness of the different materials to be crushed. This can be easily adjusted by varying the amount of bevel given to the edges of the angle iron and of the guide plate in the feed tray until the feed-stream exerts the desired uniform abrading action.

The flanges upon the longer roll-shell perform two functions. They not only confine the feed-stream at its end and prevent ridges from forming at the ends of the unflanged roll-shell, but by carrying the feed-stream a slight distance beyond the end of the shorter roll-shell, a small amount of crushing is performed between the sides of the flanges and the ends of the roll-shells. This flange crushing, being in opposite directions, balances, and all tendency to exert end thrust of the shafts against their bearings is in this manner neutralized and overcome.

By the simple means thus provided it becomes possible for the roll tender to maintain the roll-faces parallel while crushing even the hardest material, and to keep the rolls in continuous operation until the roll-shells are completely worn out. The only assistance required of a machinist is occasionally to turn off the edges of the flanges when they become so long that they strike the bolts of the opposite roll which hold its roll core in place.

This new roll design and manner of operating rolls has been developed by Mr. Frazee in connection with the dry crushing of very hard materials used for abrasives, such as quartz, garnet, etc. It is equally applicable to ores, and by the settling and removal of water from the mill feed it is possible to apply it to wet crushing as well as dry. The field which is thus opened up in connection with ores which require careful granulation, in order to prepare them for concentration, is a very large one. Mr. Payne thinks there is an especially large field for this new roll design in the treatment of complex ores, when very fine crushing without production of slimes is essential.

Mr. Payne's paper was briefly discussed by Mr. Spilsbury and Dr. R. H. Richards.

Electrolytic Copper Refining.

The first paper presented at the Wednesday afternoon session was of very pathetic interest, being the last contribution to metallurgical literature by the late HORACE H. EMRICH. As noted in our issue of November of last year, page 617, Mr. Emrich was killed on Oct. 17 at Kyshtim, Russia, where he was manager of the copper refinery; it is believed that his murderer was a discharged workman.

Early in 1911 Mr. Emrich had sent from Russia to his uncle, Mr. Karl Eilers, in New York the first part of his paper; the second and more important part was received by Mr. Eilers five days after receipt of the cable message of Mr. Emrich's death.

Mr. Emrich's paper was summed up by Dr. R. W. Raymond.

The first part of Mr. Emrich's paper dealt with the refining of copper in the reverberatory furnace for making anodes. It is shown that there is a limit beyond which it does not pay to go commercially in the refining process in the reverboratory furnace.

The second part deals with electrolytic copper refining and describes experiments which Mr. Emrich began when he was superintendent of the copper refinery of the American Smelting & Refining Company at Maurer, N. J. He wanted to find the reasons why bad cathodes are bad. His chief observation was that when he took a portion of a "bad cathode" *without re-melting it*, he could draw a wire from it that was very good and had 99.5 per cent electrical conductivity. Hence, the impurities which were in the cathode were not alloyed or chemically combined with the cathode copper, but were mechanically disseminated in it. It is only as a result of re-melting the cathode copper that the impurities become alloyed with the copper with the resultant decrease in electrical conductivity.

Now the question arises: If the impurities are simply mechanically embedded, are these impurities really precipitated at all from the electrolyte? For there is the alternate possibility that these impurities have their origin in the anode slimes and have been carried over mechanically from the anode to the cathode.

This latter view was indeed supported by the fact that an analysis of the impurities in the copper cathode and an analysis of the anode slimes showed that the proportion of arsenic to silver in both was the same.

Further, with a pure anode and a poor electrolyte a first-class cathode may be obtained because the pure anode produces no slimes.

As long as there are sufficient copper ions in the electrolyte arsenic will not be deposited on the cathode.

In the discussion Dr. Keith, Mr. Eilers, and Dr. Ledoux participated. Dr. Ledoux stated that in some cases poor results could be traced to carelessness in preparing the molds and in casting. He also told some rather peculiar cases of difficulties experienced in electrolytic copper refineries.

Decomposition Temperatures of Sulphates.

A paper by Prof. H. O. Hofman and Mr. W. Wanjukow on "the decomposition of some metallic sulphates at an elevated temperature in a current of air" was presented by Professor Hofman.

The subject is of very great practical importance in the roasting of sulphide ores and there is a regretable lack of exact data. It is true, various investigators have worked on this subject, but their results are hardly comparable with each other on account of the differences in the arrangement of the experiments.

Recently Professor Friedrich, in Breslau, has made a series of careful determinations, using a tube, one end of which was closed. Professor Hofman's arrangement differs from this in that he used a tube open at both ends. Hence there are differences between the results, Friedrich's temperatures being generally higher.

Professor Hofman used a tube open at both ends so as to have an open air current, so that SO_3 is withdrawn as soon as formed. This corresponds to actual practice conditions better than the use of a tube closed at one end.

The time required is quite different with the different materials, since some give up SO_3 readily, others do so slowly.

The determinations so far made by Professor Hofman are given in Table 1.

TABLE 1.

Sulphate	Decomposition Temperature Degree C.	Product.
$FeSO_4$	150°	$Fe_2O_3(SO_3)_2$
$Fe_2(SO_4)_3$	530°	Fe_2O_3
$Bi_2(SO_4)_3$	540°	$5\,Bi_2O_3{\cdot}4\,(SO_3)_2$
$CuSO_4$	653°	$2\,CuO\,SO_3$
$MnSO_4$	680°	Mn_3O_4
$2\,CuO.SO_3$	704°	CuO
$NiSO_4$	708°	NiO
$CoSO_4$	718°	CoO
$ZnSO_4$	739°	ZnO
$CdSO_4$	830°	$5\,CdO.SO_3$
$5\,Bi_2O_3{\cdot}4\,(SO_3)_2$	850°	Bi_2O_3
$5\,CdO\,SO_3$	875°	CdO
$CaSO_4$	1200°	CaO
$BaSO_4$	1500°	BaO

Still missing $MgSO_4$, $Al_2(SO_4)_3$, $PbSO_4$, Ag_2SO_4.

Professor Hofman's paper was briefly discussed by Dr. Ledoux and Mr. W. A. McA. Johnson, who stated that for $ZnSO_4$ he had found a decomposition temperature very close to that of Professor Hofman.

Dust Precipitations by Electrostatic Means.

The last paper of the meeting—by no means the least in interest or importance—was presented by Dr. F. G. Cottrell, of the Bureau of Mines, on his electrostatic process for the precipitation of suspended particles from gases and fumes.

Dr. Cottrell's extempore remarks on the work done by him in the past five years or so, while he was professor at the University of California, proved really fascinating. His address

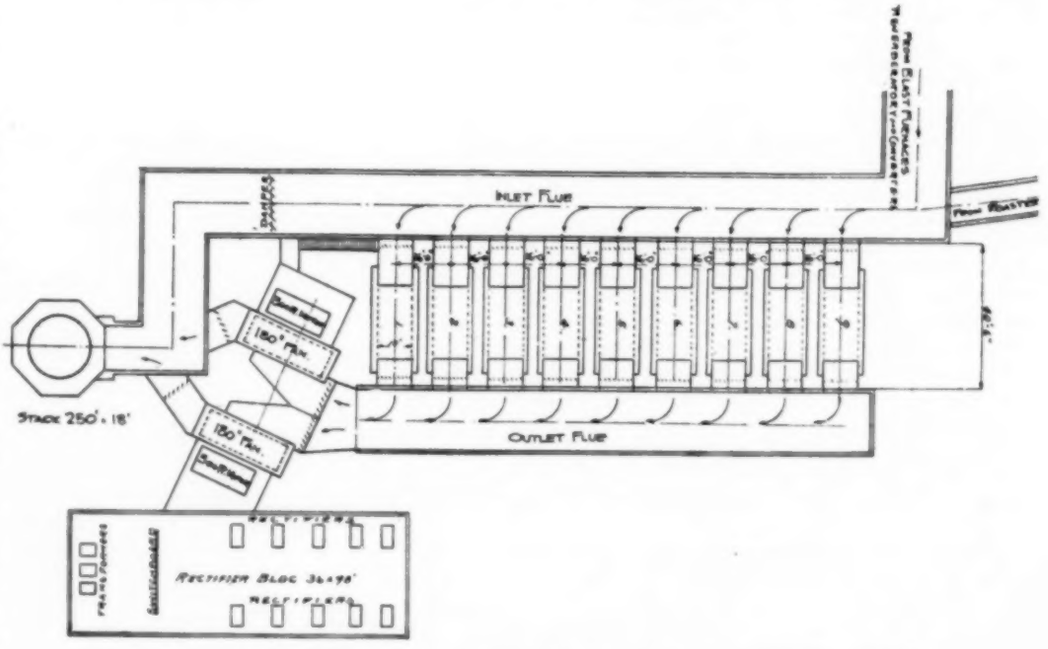


FIG. 8.—GENERAL ARRANGEMENT OF PRECIPITATION PLANT AT BALAKLALA SMELTER

was brilliantly illustrated by many very instructive lantern slides.

The fundamental principle of Dr. Cottrell's process is the same as that of Sir Oliver Lodge's early scientific pioneer work on the same subject. But Dr. Cottrell succeeded in reducing the scientific idea to successful engineering practice on a large scale.

The principle of Dr. Cottrell's process may be described by means of an old well-known experiment.

If a metallic needle point is placed opposite a flat metallic plate and the needle is connected to one pole and the plate to the other pole of a high-voltage direct-current supply line, electricity streams out of the needle point and charges the gas molecules in the space between needle and plate. The gas molecules thus receive an electric charge of the same sign as the needle point, hence opposite to the sign of the charge of the plate. They are, therefore, attracted by the plate and move toward the plate. This is the old, familiar phenomenon of the "electric wind" which may be illustrated by blowing out a candle flame brought into the space between needle point and plate.

Now if the space between needle point and flame is filled with a gas or fume in which particles of dust, etc., are suspended, these dust particles will be immediately charged with electricity and will, therefore, move toward the plate, stick to it and give up their charges. The speed of movement of the particles will be proportional to their charge and to the electrostatic field intensity (in volts per centimeter) in the space between point and plate.

This is concisely Dr. Cottrell's process. To make it suitable for working on a large scale, two fundamental problems had to be solved.

First, a practical, convenient, fool-proof method had to be evolved for the generation of a high-voltage direct current. (This is one of the differences from Lodge's work, who used the less effective alternating current.) Dr. Cottrell employs

ordinary commercial 20,000-volt to 30,000-volt alternating current and converts it into intermittent direct current by means of a specially designed rotary synchronous commutator. This intermittent direct current is directly used for charging the system of electrodes (needles and plates) in the flues which carry the gas under treatment.

The second important problem was to evolve a suitable mechanical construction of the electrode corresponding to the needle point in the above experiment. It is important to have as many and effective discharge points as possible. The solution of this problem was found in twisting asbestos filaments or mica scales with wires. The electricity passes from the wires by surface leakage over the asbestos or mica and the fine filaments of the asbestos or edges of the mica provide the required very fine discharge points, which are, indeed, far more effective than metallic discharge points.

As just explained this method presupposes that the electricity passes by surface leakage over the asbestos or mica surfaces. This surface leakage is made possible by the slight deposit of moisture or acid fume which naturally settles on the mica and asbestos surfaces. If the condition of the gases does not supply sufficiently such coating, the mica or asbestos receive a special treatment before being put into use.

The first lantern slides of Dr. Cottrell showed the arrangement of the original laboratory experiments in which the fumes of a little contact sulphuric acid plant were treated.

There was a big step from this small laboratory experiment to the large installation at the Balaklala copper smelter in Shasta County, California, shown in the next slides. This is one of the classical districts of injunctions of farmers against smelters.

The gases from the blast furnaces, roasters, etc., in the Balaklala smelter are passed into a common flue 18 ft. x 20 ft. in cross-section. The volume of gases passing through this flue vary with operating conditions for 250,000 cu. ft. to 500,000 cu. ft. a minute, the mean velocity of the fumes in the flue being between 10 ft. and 20 ft. per second.

Fig. 8 gives a plan of the nine Cottrell precipitation chambers in their relation to the flue system. Three-phase currents at 2300 volts are transformed up to 25,000 or 30,000 volts and then rectified into intermittent direct current, as noted above, and distributed to the various Cottrell precipitation chambers.

Fig. 9 is a cross-section through one of the Cottrell precipitation chambers. The double vertical lines represent the collecting or grounded plate electrodes, each 6 in. wide by 10 ft. high, made of No. 10 sheet iron.

The dotted lines represent the discharge electrodes consisting of two iron wire strands between which are twisted the asbestos and mica flakes which provide the discharge points. Each chamber contains twenty-four rows of twenty-four electrodes.

Whenever sufficient dust has accumulated on the plate electrodes, the current is cut off and the plate electrodes are shaken free from the dust, which falls down and is carried by the conveyor in each unit to a common longitudinal conveyor.

The installation is not in operation now, since by court decree the Balaklala smelter was shut down until it can live up to the letter of the original degree, viz., remove *all* solids, never exceed 0.75 per cent sulphur dioxide, and *do no damage*. The chief difficulty is in the sulphur dioxide. That the Cottrell system is successful for settling the suspended dust particles was completely proved.

Another more recent application of the Cottrell process is the settling of dust from Portland cement kiln gases, which is now in successful use at Riverside, in southern California. Since the orange groves in that district are exceedingly valuable, the prevention of dust from the cement works (which are ideally located there in every other respect) was very important. This problem has been successfully solved by the installation of the Cottrell system at the Riverside Portland Cement Works. This was shown in various lantern slides.

One amusing feature of this installation is the recovery of potash from the settled dust, so that the farmers who were responsible for the installation will also enjoy this by-product profit.

Another recent development is an experimental plant for the treatment of copper converter gases for the recovery of lead. An interesting new feature of this plant is that no flat plates are used for the settling of the dust, but round pipes with the discharge wires in the center.

Some of the lantern slides shown by Dr. Cottrell were striking in showing the smoke coming from the smokestack. This was always shown by two photographs side by side; the first was taken while the electric current was switched off and showed a big black volume of smoke; the second photograph was taken a few seconds later after the current had been switched on and showed an absolute absence of black smoke, thus proving the almost instantaneous effectiveness of the electric process.

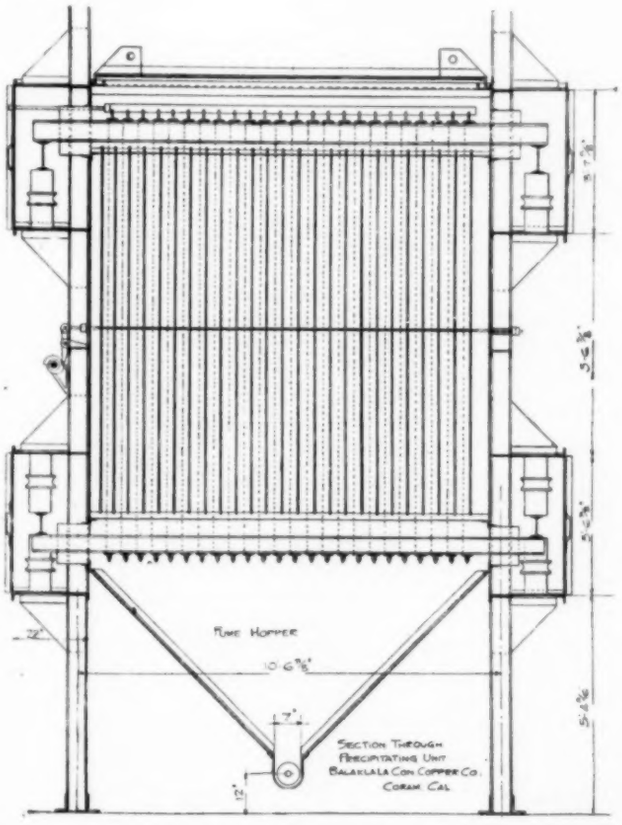


FIG. 9.—CROSS-SECTION THROUGH ONE OF THE ELECTROSTATIC PRECIPITATION CHAMBERS.

Dr. Cottrell finally referred briefly to his signing over his patent rights to the Smithsonian Institute, which will form a subsidiary corporation, the proceeds from the process to be used for the encouragement and aid of research. Considerable interest is being shown in this idealistic development not only in this country, but abroad. When the formation of the subsidiary corporation will be effected, we will cover this non-metallurgical, yet very inspiring feature of Dr. Cottrell's work in greater detail.

In the discussion which followed, Dr. Cottrell stated that the shaking off of the dust from the plates offered no difficulty whatever in practice. Simple rattling is sufficient.

As to the cost, the first cost of the installation of the Balaklala smelter, which can treat 500,000 cu. ft. of gas a minute, was about $135,000 until the construction was finished.

The first cost of the installation of the cement plant, which has about the same capacity, was less, being about $100,000.

With the more recent pipe construction adopted for the treatment of converter gases, as mentioned above, the first cost may be very materially reduced.

The cost of operation is very small. The power cost is almost insignificant. At the Balaklala smelter it was 120 kw. At the Riverside cement plant, which turns out 400 bbl. a day, it is 50 kw for the whole plant, which is negligible compared with the other power requirements of the plant.

Mining and Geological Papers.

In the Wednesday morning session Mr. S. A. Krom presented a paper on the James diagonal slimes concentrating table. It is the invention of Mr. U. S. James, of Newark, N. J., and its first installation is at the Hudson Bay Mine, in Cobalt, Ont., Canada. Since the paper was not printed in advance and the construction of the table cannot be understood without the aid of diagrams, we defer a description of the James table for a later occasion.

The following papers on mining and geological subjects were also presented and discussed at the meeting:

Prof. James F. Kemp on the Gold Hill Mining District in Western Utah.

Prof. Charles P. Berkey on the Geology of the New Catskill Aqueduct (giving some most interesting details on the geology of New York City).

Dr. F. F. Hahn on the Geology of the Salt Deposits of Western Germany.

Prof. C. A. Stewart on the Geology and Ore Deposits of the Silverbell Mining District.

Prof. Alfred C. Lane on Stagnant Mine Waters.

Mr. N. H. Emmons on A Concise Method of Showing Ore Reserves. This paper elicited quite an extended discussion as to the limits within which the proposed method may be applied in practice.

There was also on exhibition a new and very convenient electric miners' lamp made by the General Electric Company.

Before the meeting adjourned the thanks of the Institute were voted to the retiring president, Mr. Charles Kirchhoff, for all the successful work done by him during his administration.

Notes on Chemistry and Metallurgy in Great Britain.

(From our London Correspondent.)

The Standardization of Electrical Machinery.

At the meeting of the Institute of Electrical Engineers held early in December a paper was read by Dr. Robert Pohl entitled "Notes on National and International Standards for Electrical Machinery." The author reviewed the issues of the British Engineering Standards, and said the desired revision had possibly been delayed for the outcome of the deliberations of the International Electrotechnical Commission, the president of which took the hopeful view that "in many particulars the countries were already in such substantial agreement that the anticipated difficulties were much less than might at first appear." After dealing with the scope of standardization the author remarked that the points of any standard specification which were perhaps the most important, viz., those dealing with working qualities, had only been touched upon by the existing standards in regard to rise of temperature, and sparklessness, overload capacity, voltage regulation, efficiency and power-factor, and insulation all required consideration.

In the discussion, Mr. T. Hording Churton said the variety of frequencies and voltages in use presented serious difficulties to general standardization. There was, however, scope for framing rules to be generally recognized for the rating and testing of machines in which there was at present great diversity of opinion and practice.

Mr. W. E. Robson considered that the regulations of the Standards Committee were ineffectual. One of the main objects to be attained was the improvement of the standard of performance, but it would first be requisite to ensure the observation of the standards by all manufacturers. The sale in England of foreign-made machines which did not conform to such standards should be made an offence, but there was little hope of this being done while the open-market policy was maintained.

Mr. Robert Hammond said the object of the Standards Committee had been to secure efficiency with reduced cost of production, and their previous conclusions had not been of a haphazard nature. Many of the tests advocated by the author had actually been carried out on their recommendation. The committee selected the sizes of machines which most makers regularly produced and which therefore involved no extra outlay, but they also had to make the conditions of tests such as would keep the consulting engineer in check. There was one thing in the standard specifications he wished Dr. Pohl would acknowledge—that the committee had swept away the overload absurdity; for it was absurd to specify a certain rise of temperature for a machine with a given load and then require it to stand 25 per cent more.

Prof. Silvanus P. Thompson was of opinion that a six-hours' or other long run was unnecessary to ascertain the output and temperature rise. The temperature curve was quite regular, and the final figure could be easily calculated from the result of a shorter run.

Mr. H. Jones said that foreign competition did not affect British firms in regard to small standard machines, but was felt in schemes entailing the use of large machinery.

Dr. Pohl, in his reply, said he would like the standards to be reconsidered by a committee constituted similarly to that which drew up the previous specifications. He desired to shorten the duration of tests. He could not agree that overload had been swept away by the committee. The variation existing in this country in frequencies and voltages was no argument against an effort to achieve standardization. Manufacturing costs were too high in this country in this particular industry, and efforts for their reduction should be made in the direction of organization.

Corrosion Research.

The plant furnished by the Corrosion Committee of the Institute of Metals at the metallurgical department of the University of Liverpool for the study of the corrosion of non-ferrous metals was formally inaugurated in the middle of December by Sir George Muntz, who congratulated the university on conducting so important a research which had a particular interest for large seaports like Liverpool, and which had hitherto been much neglected by reason of its cost and the want of facilities for the work.

The investigation has for its especial object the observation of corrosion in brass marine condenser tubes, and will be under the direction of Mr. G. D. Bengough, the university lecturer in metallurgy.

The nature of the experiments and the proposed methods of carrying them out were laid down by the corrosion committee some months ago, and did not meet with a full measure of approval with some metallurgists and others whose opinions are entitled to consideration. It is therefore to be hoped that modifications of the original scheme have been or will be effected.

Estimation of Free Sulphur in Spent Oxide.

Before the London section of the Society of Chemical Industry, Messrs. C. Davis and J. L. Foucar recently described a quick and fairly accurate method of volumetrically determining the sulphur in spent oxide, particularly such as contains small amounts of organic matter soluble in the solvents usually employed for the extraction process, which necessarily gives results higher than the truth, while the oxidation method is prolonged and requires careful manipulation. The new method

consists in digesting the finely pulverized oxide, after drying, with sodic cyanide solution in absolute alcohol, filtering, and titrating the sulphocyanide. The results do not vary widely from those obtained by oxidation, and are accurate enough for all commercial purposes.

Engineering Imports and Exports.

The returns issued by the Board of Trade for the year 1911, put the value of imports of iron and steel, including manufactures, at £11,144,211, an increase of £2,057,997 on the year 1910, and exports at £43,751,618, an increase of £774,947. Imports of other metals including manufactures, reached £27,592,-152, showing a rise of £2,892,058, and exports £11,027,269, also a rise of £$674,915. Imports of electrical goods totalled only £1,435,429, a decline of £251,111, and exports were £2,824,271, a loss of £1,278,331. Imports of machinery went up to £5,679,-274, increasing by £1,298,376, and exports to £30,977,403, an increase of £1,706,023. Imports of new ships are valued at £64,-484, an advance of £37,380, while exports were £5,662,045, dropping £3,108,159.

Taking the month of December alone, imports increased were iron and steel, £40,209; machinery, £57,362; and the decreases were other metals, £33,271; electrical goods, £60,806; ships, £916. Exports increases: Iron and steel, £111,951; other metals, £78,751; machinery, £87,850; ships, £645,538. Exports decrease: electrical goods, by £388,059.

Market Prices.

January, 1912.

Copper has not been so steady as appeared probable. Starting at £63 it rose by the 8th to £64⅛ and then fell (15th) to £60¾, rising again to £62 (19th) it has since declined and closes £61.5s.0d. At present the fall in price of this metal is a leading feature in the metal market.

Tin dropped sharply from opening price of £200 to £190 on the 3d, recovered to £193 by the 5th, and then fell to £183 (12th), since rising to £191 (18th) and closing at £194.

Lead has been fairly steady with slight weakening 12th to 19th. Opened at £15, closes at £15-17-6.

Haematite has been uneven. Opened at 64/9, fell to 62/10 by the 11th, recovered to 63/6 by the 17th, and closes at 63/6.

Scotch Pig opened 55/4, fell to 54/2 by the 10th, and remained below 55/-, closing at 54/9.

Cleveland followed the same course as Scotch Pig, opening at 49/3 and closing at 48/9.

India-rubber has not presented any remarkable features during the month. Opening at 4/3½ it closes at 4/8.

	£ s. d.
Alum, lump, loose, per ton	5.15.0
Antimony, black sulphide powder	20. 0.0
Borax, British refined crystal, per ton	16.10.0
Copper, Ore, 10 to 25 per cent per unit	11/- to 11.6
Copper Sulphate, per ton	23.15.0
Carbolic Acid, liquid, 97/99% per gal	1.9
Caustic Soda, Ash 48% ordinary per ton	5.10.0
Creosote, ordinary good liquid, per gal	2⅞
Hydrochloric Acid, per cwt	5.0
Mica, small slab, original cases, per lb	6d. to 2.0
Naphtha Solvent, 90% at 160 deg. C. per gal	2.6
Quicksilver, per bottle	8. 5.0
Sal-Ammoniac, lump, firsts, delivered U. K. per ton	42. 0.0
Sulphate of Ammonia, f. o. b. Liverpool, per ton	14.13.9
Sulphur, recovered, per ton	5. 0.0
Shellac, standard T. N. Orange, spot, per cwt	4. 4.0
Tin Ore, 70% per ton	£120 to 122. 0.0
Zinc, Vieille Montagne, sheets	30. 0.0

Differences.

Rise.

	£ s. d.
Alum	2.6
Borax	10.0
Copper ore	3
Copper sulphate	15.0
Quicksilver	2.6
Sulphate of ammonia	8.9
Haematite	6
India rubber	5

Fall.

	£ s. d.
Copper	2. 5.0
Tin	12. 0.0
Lead	5.0
Haematite	2.6
Scotch pig	1.9
Cleveland	11

Synopsis of Metallurgical and Chemical Literature.

Iron and Steel.

Reduction of Iron Oxides in the Blast Furnace, Open Hearth, and Graphite Crucible.—In *Stahl und Eisen*, 1911, November 2, page 1808, a brief abstract is given of a paper by the Russian author, B. POMERANZEFF, who has come to some conclusions on this subject quite different from the views ordinarily accepted.

1. The reaction common assumed to be

$$FeO + CO = CO_2 + Fe$$

is considered not to be correct. It should be replaced by the following equation:

$$3\,FeO + CO + C = 2\,CO_2 + 3\,Fe$$

2. Reduction by means of elemental carbon (uncombined) in a blast furnace, open hearth, or graphite crucible does not occur.

3. For the reactions or processes which occur in a blast furnace the following equations are given:

$$Fe_3O_4 + CO = CO_2 + FeO$$
$$3FeO + CO + C = 2\,CO_2 + 3\,FeO$$
$$3\,FeO + CO + 2C = 2\,CO_2 + Fe_3C$$
$$Fe_3C + FeO\,CaO\,SiO_2 = 4\,Fe + CO + CaO\,SiO_2$$
$$3\,Fe_3C + MnO\,SiO_2 = 9\,Fe + Mn + Si + 3CO$$

No explanation of these equations is given in the German abstract, the original paper having appeared in the *Journal of the Russian Met. Soc.*, 1911, Vol. 4, p. 366/75.

Smelting and Reduction of Titaniferous Iron Ores in Electric Furnaces.—Prof. W. BORCHERS discusses in *Stahl und Eisen*, May 4, 1911, page 1706, the various investigations carried out on this subject in his laboratory at Aachen. When an oxide ore is smelted two layers must be usually distinguished, the lower layer consisting of the heavier oxides and the upper layer being the slag which results from the gangue. If for some reason this process has to be conducted in an electric furnace and the oxide is to be reduced to metal by means of coke, charcoal, etc., an undesirable phenomenon is liable to happen. Before the reducing agent becomes effective the heavy oxide may separate from the mixture and accumulate on the bottom of the furnace (perhaps with some reduced metal below), while the reducing agent is separated from the oxide by the liquid slag. (This being called a case of "liquid concentration.") Nor does occasional agitation prove a remedy. Briquetting of ore and coal previous to the smelting process would result in an increase of cost and require separate apparatus.

In Borchers' laboratory tests for smelting titaniferous ores have been conducted for about twelve years for the purpose of reducing the iron and recovering the titanium in form of a useful product. Huppertz worked along these lines in 1904 on the reduction of titanic acid and compounds. Further Lugner (1898-99) and Huppertz (1904) showed that the reduction of titaniferous ores in an electric furnace offers no

special difficulties, iron with small percentage of titanium being reduced, while most of the titanium is concentrated in the slag. It was shown that, in the absence of nitrogen and at temperatures of below 2000° C., only a very small amount of titanium is reduced into the iron. The blast furnace where such temperatures prevail is, however, not a very successful instrument for this process owing to the formation of nitrides and cyanides of titanium and iron on account of the great quantities of nitrogen present in the air of the blast.

The first tests by Borchers on melting a mixture of titaniferous ores with reducing agents did not turn out very satisfactory owing to the phenomenon of "liquid concentration" mentioned above. The power consumption per unit of output was exceedingly high, while even agitating would not bring the molten ore in contact with the carbon. Another remedy had to be looked for and was found in the following fact:

If titaniferous ore is smelted with an insufficient amount of carbon in order to reduce iron oxide a part of the iron oxide will be consumed for the formation of slag with titanic acid. With just the right amount of coal, a fairly liquid, highly titaniferous acid slag is obtained, its composition being within the limits: 40-47 per cent TiO_2, 20-5 per cent SiO_2, 34-30 per cent FeO. This titanate, as it may be called, may be used as the raw material for the production of titanium alloys. When reduced by carbon a mixture of iron-titanide, titanium carbide, and metallic titanium is obtained with free carbon according to the amount of carbon used, and consisting approximately of 42-48 per cent Ti, 5-1 per cent Si, 20-6 per cent C, 30-8 per cent Fe. This mixture of titanides and carbides is a very good reducing agent; moreover, it is heavy enough to come in direct contact with the molten ore. In other words, the reducing carbon is burdened with iron, titanium, and silicon, to produce a very efficient reducing agent, by which means the time for the main reaction between ore and reducing agent is considerably reduced while the power consumption becomes a minimum.

A scheme for a large plant using this process might be as follows: For the production of 1000 kg of iron two furnaces are to be used simultaneously.

Furnace 1—Electric resistance furnace: 400-450 kg carbon; 1000-1100 kg slag of above composition.

Furnace 2—Gas or electric furnace. 1600-1700 kg titaniferous ore containing 50 per cent Fe, 10-15 per cent TiO_2.

The molten material from furnace 2 is poured into furnace 1. Furnace 2 may be constructed larger than furnace 1, so that only part of its contents has to be added into one or more furnaces of type 1. It would hardly be preferable to reverse this practice, since the material in furnace 1 will not be melted and it is better to keep it at a very high temperature. The reaction in furnace 1 after addition from furnace 2 will not require any power consumption in larger furnaces and will give as product about 1000 kg iron and 1500 kg slag.

From this slag 1000-1100 kg are returned to furnace group 1 together with carbon; the rest of 400 kg is available for production of alloys of titanium. The flue gases of furnace 2, if operated by gas, may be utilized again in some way.

Nissen Stamps on the Rand.—At the City Deep mill on the Rand a comparative test between Nissen single-mortar stamps and ordinary stamps has been in progress. In the *Journal* of the Chemical, Metallurgical & Mining Society, of South Africa, for Oct. 1911, Mr. Peter N. Nissen describes the advantages of the Nissen stamp and gives comparative results of the test mentioned. The Nissen stamp is of the heavy type, having been used as heavy as 2250 lb. It has a single mortar with circular screen, and larger diameter of shoes and dies than is common in heavy stamps, viz., 10 in. For the same tonnage the Nissen stamp requires about 30 per cent less power than ordinary stamps, and 40 per cent less weight of iron and steel in the battery. The accompanying tables embody the essential features of the test.

Continuous Decantation Accessory to Vacuum Filtration.—The use of Dorr thickeners in continuous decantation schemes has been suggested frequently. In the *Informes y Memorias* of the Mexican Institute of Mining and Metallurgy, Vol. 2, No. 5, J. Leslie Mennell suggests their use in connection with vacuum filters, as a means of removing a large portion of the soluble precious metals from the pulp before sending it to the filters. He has been working on specially slimy ores which do not readily yield to vacuum filtration, and has been bothered by excessive losses of soluble silver in the filter cakes. To give a concrete example: The soluble value left in cakes formed from pulp in which the solution carried 200 grams

Table No. I.—Summary of Comparative Tests between Nissen and "City Deep" Stamps.

Test Number and Date	1 June 23, 1911		2 July 5, 1911		3 July 17, 1911		4 July 26, 1911		5 Sept. 11, 1911		6 Sept. 16, 1911	
No. of Stamps and kind	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.	FOUR NISSEN.	TEN CITY DEEP.
Weight of Stamps in lb.	1932	1863	1927	1859	2245	1855	2237		1993	1775	1991	1773
No. of Drops per min.	103	100	103	100	103	100	103		103	100	103	100
Height of Drop in inches	8½	8½	8½	8½	8½	8½	8½		8½	8½	8½	8½
Height of Discharge	2½	2½	2	2½	2	2½	2		2½	2½	2½	2½
Class of Screen	Tyler	T.C.	Tyler	T.C.	Sq. m.	14 (.08)	T.T.C.		Sq. m.	wire	Sq. m.	wire
Mesh of Screen	9	9	9	9	⅜"	⅜"	9		9	9	⅜"	⅜"
Width Aperture in inches	.205	.205	.205	.205	.375	.375	.23		.277	.277	.375	.375
Length Aperture in inches	.536	.536	.536	.536	.375	.375	.85		.277	.277	.375	.375
Dry Ore crushed tons	180.86	192.75	180.25	192.12	193.88	206.66	191.7		185.32	194.53	185.7	191.2
Total Run in hours	44.33	25.33	39.0	23.166	31.70	21.83	37.28		37.29	22.28	29.52	18.85
Tons per stamp per 24 hr.	24.47	18.26	27.73	19.90	36.69	22.72	30.85		29.81	20.95	37.74	24.34
H.P. per stamp*	4.09	4.21	4.09	4.21	4.95	4.21	4.95		4.23	4.65	4.23	4.65
H.P. hours per ton of ore crushed*	4.0	5.5	3.56	5.06	3.24	4.45	3.86		3.41	5.32	2.71	4.56
Llbs. ore crushed per lb. falling wt. per day	25.33	19.6	28.78	21.412	32.68	24.49	27.58		29.91	23.6	37.91	27.45
Ratio water, ore wt.	5.4 : 1	2.7 : 1	4.2 : 1	2.4 : 1	3.1 : 1	2.15 : 1	3.2 : 1		3.3 : 1	3.3 : 1	3.03 : 1	3.26 : 1
Cons. metal in shoes in lb. per ton crushed.	.129	.145	.139	.24	.198	.187	.122		.147	—	.112	.08

NO COMPETITIVE TEST

*Excluding Belt, Shafting and Motor losses.

Grading Analysis.

Feed	1		2		3		4		5		6	
+2	36.9	12.1	14.96	12.4	26.9	18.2	52.3		39.0	14.1	23.7	27.1
+1½	18.6	14.4	14.50	11.35	18.75	16.0	14.5		19.7	18.0	20.9	17.1
+¾	15.1	20.2	20.40	19.35	19.3	18.15	10.4		15.0	21.3	18.8	19.1
+¼	11.7	12.7	16.95	17.70	13.05	16.25	9.4		9.3	16.0	12.7	12.6
−¼	27.5	37.9	33.20	39.20	22.0	31.4	13.4		17.0	30.6	23.9	24.1
Discharge												
+60	65.2	63.4	64.9	63.4	67.24	69.33	67.3		66.6	66.5	70.7	70.7
+90	8.3	9.1	8.46	8.70	8.62	9.09	8.75		9.9	10.2	10.1	10.3
−90	9.7	12.0	8.46	9.10	10.84	9.82	8.55		8.8	10.9	10.2	9.2
−200	16.7	10.7	18.10	19.5	13.3	11.76	15.4		14.7	12.4	9.0	9.8

TABLE NO. II.—COMPARATIVE STATEMENT OF CAPACITY, POWER DEMAND, CAPITAL COST AND WEAR OF SHOES AND DIES FOR NISSEN AND ORDINARY STAMPS.

	Nissen Stamp.	Ordinary Stamp.
1. Net capacity per month (tons)	40,000	40,000
2. Net capacity per day (tons)	1,330	1,330
3. Percentage of net working time to total time	95%	90%
4. Gross capacity per day (tons)	1,400	1,480
5. Size of screen. (9 mesh per square inch)	0.277	0.277
6. Falling weight of stamp (pounds)	2,000	1,480
7. Duty per stamp per day of 24 hours (tons)	29.16	14.8
8. Number of stamps required	48	100
9. Power consumption per stamp, hp	4.3	3.02
10. Power consumption of entire mill, hp	206.4	300
11. Loss of power in motors, shafting and belting (a) Nissen mill, 5 drives at 7.8 hp, 2 drives at 4.8 hp	48.6	
(b) Ordinary mill, 10 drives at 7.8 hp		78
12. Total power demand, hp	255	370
13. Total power demand, kw	190	276
14. Cost of power per kw-hour (pence)	0.6	0.6
15. Cost of power per day	£11 7s. 10d.	£16 11s. 2d.
16. Saving in favor of Nissen stamps per day	£5 3s. 4d.	
17. Saving in favor of Nissen stamps per ton milled (pence)	0.931	
18. Capital cost of battery	£20,000	£23,000
19. Capital cost of battery (per ton milled per day)	£15.04	£17.29
20. Saving in favor of Nissen mill	£3,000	
21. Interest on capital saved at 12% per annum, per month	£30	
22. Interest on capital saved at 12% per day	£1	
23. Interest on capital saved per ton milled (pence)	0.180	
24. Wear on shoes and dies per ton milled (lb.)	0.3	0.4
25. Cost of shoes and dies at 3d. per lb. per ton milled (pence)	0.6	0.8
26. Difference in favor of Nissen mill (pence)	0.2	
Total saving (18+20+931), pence per ton milled	1.311d.	

silver per ton, has been about 30 grams silver per ton of cake, despite a wash of barren solution for 45 minutes and water for 10 minutes.

He proposes a scheme whereby Dorr thickeners installed ahead of the vacuum filter shall remove the greater part of his rich solution by decantation, so that the pulp finally going to the filter shall be comparatively poor, and the loss in the cake correspondingly low. The ore is crushed and ground in cyanid solution, agitated in Pachuca tanks at a consistency of 1.5:1, and sent to a thinning tank where mill solution is added, giving a consistentcy of 6:1. The solution now carries only 76.5 grams silver per ton as compared with 162 grams as it came from the Pachucas. It is now thickened in a Dorr machine and rich solution drawn off carrying 68.5% of the silver. This is the only solution that is precipitated, and effects the immediate removal of 68.5% of the value. The thick pulp issuing from the Dorr is again thinned with barren solution, and thickened in a second Dorr, whereby additional silver recovery is effected, bringing the total removal up to 94%. The thick pulp from the second Dorr is now sent to the Butters filter, carrying only 6% of the original silver value. Here it is filtered and washed as effectively as possible. Granting only a 50% displacement, there is recovered an additional 4.5% silver, making a total of 98.5%. The cake discharged will contain but 3.6 grams silver per ton of dry ore, instead of 30 grams as by direct filtration.

Recent Metallurgical Patents.

Gold and Silver.

Reduction of Arsenical Silver Ores.—The silver ores occurring in the Cobalt district, Canada, have been the subject of much experimenting to find suitable methods of recovering the cobalt, nickel and arsenic which they contain in appreciable quantities. Mr. COLIN G. FINK, of Schenectady, N. Y., has recently patented a combination process of treating these ores, which he assigned to the General Electric Company. A typical example is as follows: The ore contains, 9.7 per cent cobalt, 4.65 per cent nickel, 28.65 per cent arsenic, 18.06 per cent silver, with the balance of 39 per cent composed of iron, aluminium, calcium, silicon, antimony and perhaps other elements. The ore is ground to 40-mesh size, fluxed in a suitable manner to reduce the metals and form a slag of the gangue, and charged into a crucible or furnace where it can be heated under reduced pressure, as in the Arsem furnace (patent 785,535, 1905). The temperature is raised to 1200-1500 deg. C., and maintained for two or three hours. By this treatment the greater part of the arsenic is volatilized and condensed as metallic arsenic, and the charge in the crucible separates into strata of metallic silver, speiss of cobalt and nickel, and slag in the order named, silver being the lowermost. The charge can be allowed to cool and the separation of the strata made mechanically, or the silver can be tapped and purified by distillation in an Arsem furnace. The speiss also can be separated and treated in known manner for recovery of the cobalt and nickel. (1,013,931, Jan. 9, 1912.)

Imlay Apparatus for Cyanidation.—Claims of economy in cost of plant and operation are made for certain apparatus recently patented by WILLIAM L. IMLAY, of Philadelphia, Pa. By his method, the ore is not classified into sand and slime until after lixiviation in his machine. The apparatus consists of a slowly rotating cylinder, at one end of which the ore pulp in cyanide solution is fed, being discharged at the other end. The cylinder is lined with a helical flange and ribs which act as buckets. The cylinder is about 50 ft. long and makes 1 to 5 r.p.m. As it revolves the ore is slowly conveyed from one end to the other by the helical flange, and is continuously raised by the ribs and allowed to fall back into the mass of the charge. It is the inventor's claim that this agitation and aeration are most effective, and that a saving is effected in cost of plant and operation. (1,014,224, Jan. 9, 1912.)

Agitator and Classifier.—In Fig. 1 is illustrated a combination agitator and classifier with filter-bottom, which is the invention of J. W. FAIRFIELD, of Goldfield, Nevada. The inventor states that it may be used for the regular treatment of ore, but that it embodies features which make it specially

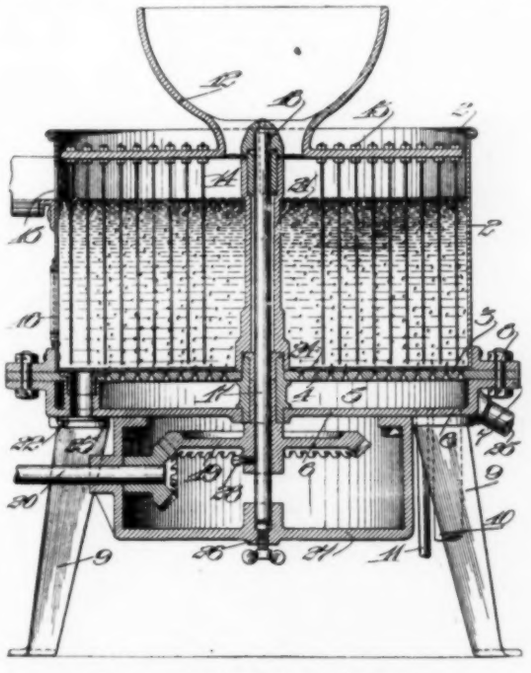


FIG. 1.—AGITATOR AND CLASSIFIER.

adapted to experimenal testing of ores by the cyanide process. It consists of a cylindrical tank 2, perforated false bottom 3, chamber 5, driving gear as shown, and feed hopper 12 to which are attached stirrers 14. An overflow spout for slime is provided at 15, and a gate for discharge of sand at 22. Water-pipe connection with the chamber 5 is shown at 10, and a similar air-pipe connection at 11. The pulp in cyanide solution is fed at the hopper 12, and falls into the tank where it is

agitated by the stirrers 14 attached to the turn-table 13. Further agitation is effected by forcing water or solution or air through the chamber 5 and false bottom 3. The process may be continuous or intermittent, slime being discharged at 15 and sand at 22. In making tests on small lots of ore, the false bottom may be covered with a filtering medium, and after agitation and aeration have proceeded for a sufficient length of time, the solution may be filtered from the solid portion by applying suction at the pipe 11. (1,010,244, Nov. 28, 1911.)

Copper.

Bradley Process for Low Grade Ores.—Considerable interest attaches to the Bradley process because it is being tested experimentally on the slime residue at the Anaconda Copper Mining Company's concentrator at Anaconda, Mont. The method is the invention of Charles S. Bradley, of New York City, and is designed to handle successfully those low grade copper ores in which a small percentage of copper must be separated from a large percentage of gangue. The Anaconda slimes carry about 2.5 per cent copper, and are therefore comparable to the porphyry ores as regards copper content, although in physical condition they are much finer and more difficult to treat by physical methods. A diagram of the Bradley chemical process is given in Fig. 2.

The chemical principles involved in the process are as follows: Calcium chloride is used as a solvent and chlorine carrier, converting copper sulphate into copper chloride, and precipitating calcium chloride. The chlorine compound thus has the advantage of forming an insoluble compound in the reaction, and may be removed. The copper chloride solution may be precipitated by the addition of calcium carbonate, whereby calcium chloride is regenerated and copper oxide precipitated with evolution of CO_2 if the temperature is high enough. It will be apparent that the reaction of calcium chloride on the copper compounds requires the presence of an acid radical, and that sulphuric acid might be used as an added reagent to bring about this effect. But the inventor prefers to produce this acid radical from the ore by first roasting it to sulphate, thereby avoiding the expense of separate addition of acid. To operate the process it is only necessary to bring in a small quantity of calcium chloride to start the reactions and make up the waste as chlorine is retained in solution, and regenerate the calcium chloride when the copper value is precipitated.

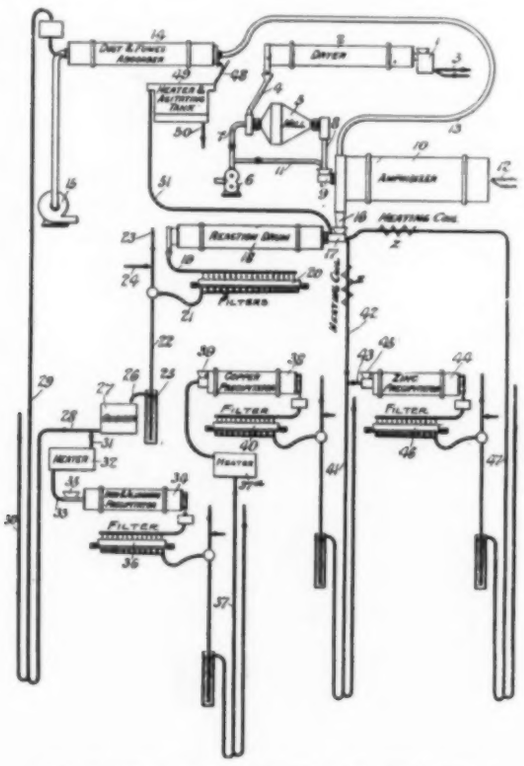


FIG. 2.—BRADLEY PROCESS FOR LOW GRADE ORES.

The inventor makes use of the term amphidizing to indicate the process of roasting copper sulphide to sulphate, and speaks of the roasting apparatus as an amphidizer. Amphide was a term used by Berzelius to distinguish those salts containing oxygen from the corresponding unoxidized salts.

The description of the process will be understood by reference to the figure. The first step is to crush and dry the material to be treated. It is then heated to a temperature of from 450 to 550 deg. C. with access of air, in the amphidizer where the sulphide of copper is largely converted into sulphate. In practice most of the iron in the ore will be converted into ferric oxide which is insoluble; but some iron will form ferric sulphate, and some of the copper will remain as sulphide, while other small portions will form copper oxide or remain as metallic copper if such was in the ore. The amphidizing apparatus does not require strict supervision as provision is made later for the treatment of any copper which shall not have been converted into sulphate, and of iron which has been formed into sulphate.

The dust and fumes which are formed in the amphidizer are collected in an absorber and made to serve a useful purpose, as will be shown.

From the apmhidizer the roasted ore passes to the reaction drum, which is rotatable and supplied with a solution of calcium chloride preferably at a little below boiling temperature. The ferric sulphate which may have been formed in the amphidizer here reacts with calcium chloride, forming ferric chloride and calcium sulphate. The ferric chloride thus formed serves to dissolve the copper oxide, sulphide and metallic copper remnants which were not oxidized in the amphidizer. Here also any sulphuric acid which may have been formed in the absorber, combines with calcium chloride to liberate hydrochloric acid which, in turn may also dissolve any unoxidized copper compounds, converting them into chlorides. The net result is that all the copper is converted into chloride, while the calcium is precipitated as sulphate. It is essential that an excess of calcium chloride be present, and that all the solutions be kept at their highest state of valency.

The pulp is now ready for filtration, and is discharged to the filters from which the calcium sulphate and gangue are eventually removed. The solution is then preferably subjected to further oxidation, as by blowing air through it, when it is ready for the precipitation and separation of the various metallic contents. It may contain iron and aluminium, as well as zinc, if such were in the ore. Calcium carbonate may be employed as the cheapest efficient substance, but inasmuch as it will precipitate iron and aluminium, as well as copper, and as copper oxide will react with iron and aluminium compounds as soon as formed, the former must be completely precipitated before the latter can remain as a precipitate and be recovered. On the other hand the iron and aluminium are useful in the absorption of the fumes, so only a part of the solution is subjected to precipitation and the other part is diverted to the absorber.

The precipitation of iron and aluminium is accomplished at the point designated in the diagram, and any copper which precipitates with them may be recovered by returning the precipitate to the amphidizer. Following the separation of the iron and aluminium, the copper is precipitated by calcium carbonate, thereby regenerating calcium chloride which is separated by filtration from the copper precipitate. If the solution contains zinc worth recovering this may be accomplished by precipitation with burnt lime. The solutions are heated prior to precipitation as indicated in the diagram. The regenerated solution is now returned to the reaction drum.

That portion of the original solution which was diverted to

the absorber reacts with the dust and fumes which have been collected, thereby recovering the values which might otherwise be lost. The sulphuric acid there formed from the SO_2 gas precipitates some calcium sulphate and liberates hydrochloric acid which, in turn oxidizes cuprous chloride which has been formed by the reaction of SO_2 and cupric chloride.

It will be noted in this system that the calcium chloride acts both as a solvent and as a carrier, circulating through the system from beginning to end, and when regenerated is restored without change of state. The chloridizing takes place in the drum and not in the roaster, and there is no tendency to overload the system with soluble chlorine salts which are difficult and expensive to remove. (1,011,562, Dec. 12, 1911.)

MacKay Copper Process.—Another proposal to make use of a metallic chloride as a solvent for copper in ores is that of HENRY S. MACKAY, of Riverside, Cal. He proposes to use ferric chloride which he prepares as follows: Sodium chloride is electrolyzed with iron anodes, producing ferric chloride and chlorine. This solution is added to the ore and the entire pulp subjected to air agitation. His claim is that the copper will be dissolved and the ferric chloride reduced to ferrous chloride. Then upon electrolyzing the filtered solution, the copper will be deposited and the solution of ferric chloride and chlorine regenerated. (1,011,459, Dec. 12, 1911.)

Electrolytic Extraction of Copper from Ores.—The rather peculiar use of chromic acid as a copper solvent is the basis of a patent granted to HENRY K. HESS, of Philadelphia, Pa. He proposes to treat copper ore suitably placed in a tank, using an electrolyte of sulphuric and chromic acids, and carry on solution and deposition at the same time. The ore is placed between electrodes, on one of which the copper dissolved by the chromic solution is deposited electrolytically. (1,014,061, Jan. 9, 1912.)

Still another proposal in the hydrometallurgy of copper is that of ANTONY GADOMSKY, of Tiflis, Russia. Calcined copper ore or uncalcined copper slags are ground and treated in a closed tank with a mixture of two solutions containing respectively 4 per cent ammonia and 4 per cent ammonium carbonate. The net result of the reaction is the formation of copper ammonium hydrate which readily dissolves in the ammonia. As soon as the solution is saturated with copper it is withdrawn to a still where the ammonia and CO_2 are driven off and dissolved in water for further use as a solvent of copper. Copper oxide is precipitated in this distilling process, and can be removed and reduced to copper by smelting. (1,014,868 Jan. 16, 1912.)

Zinc.

The Saeger Muffle Charging and Cleaning Machine.—In the illustration Fig. 3, is shown one design of the Saeger mechanical device for charging and cleaning zinc retorts. It

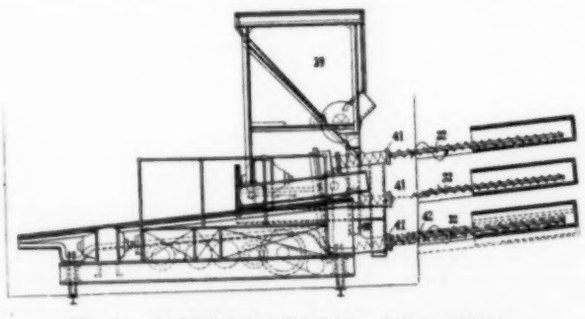


FIG. 3.—MACHINE FOR CHARGING ZINC RETORTS.

is the invention of OTTO SAEGER, of Breslau, Germany. Screw conveyers are mounted on a truck which can be moved along the front of a bank of retorts, and also in the longitudinal direction of the retorts. For this purpose a lower truck runs on tracks laid parallel with the front of the furnace, and an upper truck moves on inclined tracks laid at right angles to the front of the furnace. A charge hopper 39 is provided from which ore is fed by a distributing pipe 40 to short pipes 41 from the screw conveyers issue. In charging the retorts the screws have one direction of rotation, and in cleaning they have the reverse direction. In order that the ends of the retorts may not be damaged by reason of one screw reaching the end

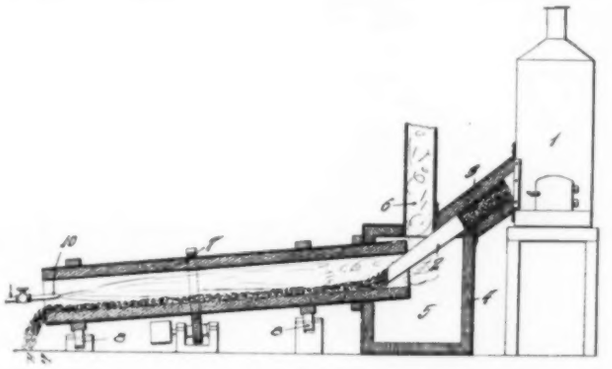


FIG. 4.—ZINC WHITE APPARATUS.

before the others, flexible connections are inserted between the conveyers and their driving shafts. (1,014,333, Jan. 9, 1912.)

Manufacture of Zinc White.—The continuous production of zinc oxide, or zinc white from roasted zinc sulphide ores may be accomplished according to the specifications of a patent granted to HARRY H. HUGHES, of St. Louis, Mo. In Fig. 4 we illustrate apparatus in which he proposes to carry out his invention. A roasting furnace is represented at 1, from which the roasted ore at a temperature of about 900 deg. C. is continuously discharged through the chute 3 into a revolving cylinder 7. The ignition of the hydrocarbon issuing from the nozzle 10 in the lower end of the cylinder brings the temperature up to approximately 1300 deg. C., which heat is necessary for the proper reduction of the ore to produce zinc white. This hydrocarbon flame acts as a reduction material, and the air drawn into the cylinder by the suction produced by the flame, together with the carbon dioxide produced by the combustion of the oil or gas, causes a reoxidation of the ore, thereby producing zinc white. Thus the zinc white is produced without adding any carbon or other reducing substance to the ore as in present methods. The zinc white fume passes out through the stack 6 and is condensed in suitable manner. Where carbonate, silicate or oxide zinc ores are used, no roasting is necessary, but a small addition of carbon may be mixed to give the proper reducing conditions. (1,014,062, Jan. 9, 1912.)

Bisulphite Process.—A modified form of the bisulphite process for zinc and copper ores is proposed by WOLDEMAR HOMMEL and HENRY T. DURANT, of London, England. Roasted zinc ore is mixed with water and brought in contact with sulphurous acid gas. The reaction forms zinc bisulphite, which is soluble and can be separated from the gangue of the ore. The solution is then subjected to the action of hot roaster gases, with two results: the gas is cooled so that it is available for the treatment of further ore and water; and the zinc bisulphite solution is broken up, giving sulphurous acid gas which enriches the furnace gas, and zinc monosulphite is precipitated. This is then collected and calcined to produce zinc oxide. The process has the commercial advantage of utilizing the heat of the roaster gases to precipitate the monosulphite, and of cooling and enriching the furnace gases for the treatment of ore. If the process is applied to copper ores, metallic copper will be precipitated instead of zinc monosulphite. (1,009,750, Nov. 28, 1911.)

Treatment of Refractory Zinc-Lead Ores.—A scheme for dissolving roasted refractory ores in zinc sulphate solution has been proposed by PERCY C. C. ISHERWOOD, of Leytonstone, England. There are some points in the process in which it seems as though mechanical difficulties would be encountered, but

otherwise the proposal is of interest. The inventor has discovered that the solubility of zinc oxide in zinc sulphate solution is greatly increased at temperatures above the boiling point of zinc sulphate and at pressure of from 50 to 200 lb. per square inch. As a result of this he finds that he can extract all the zinc from a mixed ore without contaminating his solution with iron. He proposes to roast ore so that a maximum of sulphate is formed, with some oxide and a minimum of sulphide. The roasted ore is then treated under conditions of temperature and pressure noted above, in a lead lined or enameled vessel, with less than the theoretical quantity of sulphuric acid necessary to dissolve all the zinc under normal conditions. The zinc sulphate formed is a good solvent for the oxide present, and if separated from the residue while still under pressure, and then allowed to cool, he finds that the excess of zinc will precipitate as basic sulphate. (1,011,962. Dec. 19, 1911.)

"Atomized Metals."

The manufacture of metals into a fine state of subdivision has been carried on in various industries for many years, but as the object aimed at was generally to obtain the metals in a condition readily susceptible to rapid oxidization, as for example, in the making of white lead pigment, etc., it required new conditions, etc., to develop a method by which the metals are obtained in an extremely fine state of subdivision, but unoxidized and in such a physical condition that even on exposure to ordinary atmospheric conditions they remain unoxidized. This new condition has been called *"atomized"* and the new process by which this condition can be obtained will be described in this article.

Atomized metals obtained by this process, are in such a physical state of activity, that their action in many industries may be strikingly likened to the action of ozone, while the same metals prepared by other methods are in their action similar to oxygen.

One industry to which atomized metals are especially adapted, is that of recovering the precious metals from cyanide solution obtained during the treatment of the ores by cyanide, and their application to this industry, marks a distinct and important advance in this art, through the elimination of the numerous imperfections and disadvantages existing under present conditions.

In order to understand the adapatibility of atomized metals to the art of cyaniding it will be well to give a short résumé of the numerous processes for the precipitation of precious metals from cyanide solutions; among these processes the following are those best known and in use throughout the world.

Zinc Shavings.—This process, one of the oldest—consists in converting metallic zinc into thin films or shavings, in which form it is placed in boxes through which the pregnant cyanide solutions are then allowed to percolate, with the resultant precipitation of the precious metals by replacement.

Lead-Zinc.—This process, invented by J. K. Betty, was used to some extent in South Africa with varying results. Its success depends upon the formation of an electro-galvanic couple by means of lead and zinc. This is accomplished by immersing strips of metallic zinc in a lead salt solution, usually lead acetate, whereby there is produced a deposit of metallic lead on the surface of the zinc, and this lead-zinc is then used in a manner similar to zinc shavings.

Zinc-Dust.—Prominent in the art of precipitation of precious metals from cyanide solutions stands zinc dust, its method of application or addition to the cyanide solution being the subject of numerous patents. Its efficiency is partially due to a small percentage of metallic impurities (usually lead), which have been carried over mechanically during the distilling operation, and as the always present zinc oxide is inert, it plays no part in the precipitation, being in fact detrimental, as it contaminates the precious metal precipitate and the solutions.

Zinc dust is the "blue powder" of the smelter, unavoidably formed in the distillation of the spelter. Its preparation consists merely in collecting the blue powder from the prolongs, as free from admixture with foreign material as possible, and then sifting. As its value depends upon the percentage of metallic zinc (zinc oxide being detrimental) it is essential in preparing it for this market to regulate the distillation and condensation of the zinc so that there will be the least possible formation of oxide. The market standard is a tenor of at least 90 per cent metallic *zinc*, but it is seldom as pure as this, the percentage of oxide being usually higher than 10 per cent.

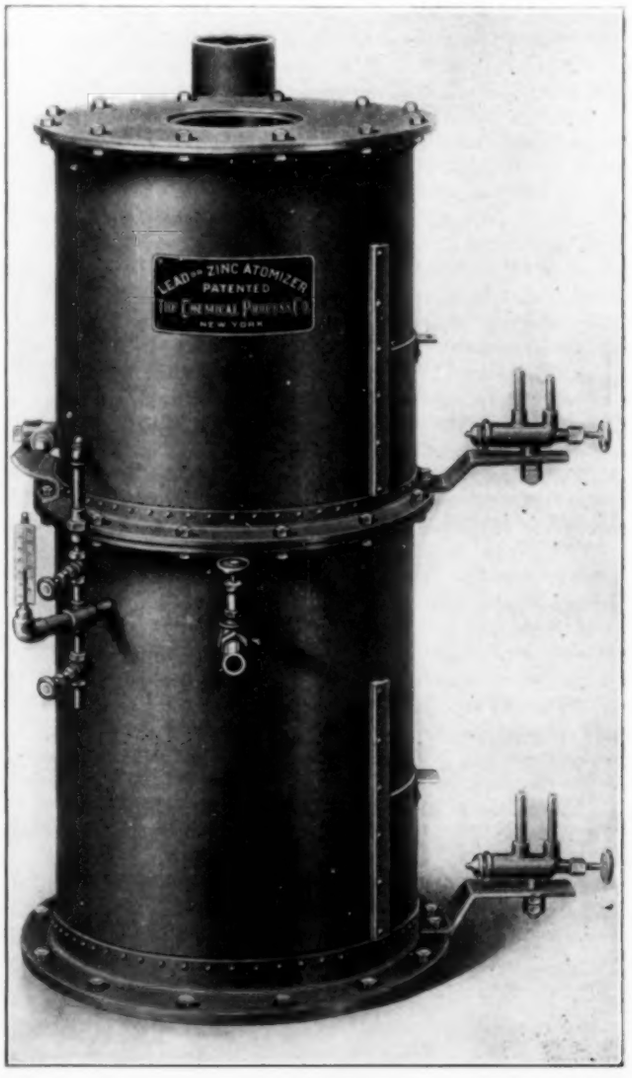


FIG. 1.—ATOMIZER.

Zinc dust is extremely susceptible to oxidation—so much so as to be explosive—and on that account great care and precaution is necessary in its transportation and storage. Owing to its inflammable nature and liability to spontaneous combustion it is rated by fire underwriters as a specially hazardous risk.

A new ruling of the Interstate Commerce Commission lists zinc dust and zinc flue dust as inflammable solids and directs that they must bear a yellow label. Paragraph 1834 of the ruling covering this product, reads as follows: "Packages must be tight and strong, and the interior packages must be so cushioned and secured that no rupture of either package can result from the ordinary shocks incident to transportation."

Notwithstanding the above mentioned disadvantages, the use of zinc dust in the process of precipitation has been preferred

to zinc shavings or the Betty lead-zinc process for the following reasons:

(a) The comparative cost of zinc dust and zinc is about 20 to 25 per cent in favor of zinc dust.

(b) Increased efficiency and decreased operative costs.

(c) In practice the consumption of zinc dust is not as great as that of zinc shavings.

(d) The re-solution of the precipitated precious metals when using zinc dust can be prevented by taking proper precautions.

(e) The system of using zinc dust minimizes the risk of loss of the precious metal precipitates from theft, fire, etc., permits of regular "clean-ups," conversion of the precipitates into bullion at any time and of accurate checks on the mill recoveries.

Having thus seen how great the advantages are of zinc dust over zinc shavings and lead-zinc, it only remains to show how preferable is the use of "atomized" lead and zinc over zinc dust. The superiority of atomized metals can be summed up as follows:

1. The production of atomized unoxidized lead and zinc, at will at a nominal cost, gives precipitating agents from which the maximum efficiency can be obtained.

2. The elimination of delays in delivery and of the dangers from spontaneous combustion and from explosion during transportation, storage, etc., and of the necessarily excessive freight charges when zinc dust is used.

3. Absolute control in the operation of precipitating the precious metals.

4. The elimination of the loss of zinc and of the contamination by oxide of the precious metal precipitates and cyanide solutions, as is the case in zinc dust.

5. The production and use of lead and zinc atomized of the same purity as the metals from which they are made.

6. The fact that pig lead and zinc from which atomized lead and zinc is produced are readily obtainable in any market.

7. The securing of any desired or required electro-galvanic action by the addition of the necessary amount of atomized lead to the atomized zinc.

8. The increased activity, efficiency and energy of the precipitating action of atomized lead and zinc is much greater than that of zinc dust.

9. The scrap zinc and lead usually found around the mill is readily converted into valuable atomized lead and zinc and thus profitably utilized instead of being disposed of as scrap metal.

10. The complete precipitation of the precious metals by the use of atomized metals.

11. The absence of the oxidization of atomized metals, as is the case with the use of zinc dust.

12. There being no absorption of water by atomized metals, the difficulties attending zinc dust are thereby lessened.

13. In using zinc dust experience has demonstrated that it is necessary to exclude the air, otherwise oxidation and secondary reactions occur, whereas in using atomized metals the exclusion of air is immaterial, as they can be used in any desired manner.

14. There is less consumption of acid in the final treatment of the precious metal precipitates. Microscropical examination shows that in zinc dust the particles of metallic zinc are surrounded by a film of oxide, which prevents any precipitating action, while the surfaces of the particles of atomized metals are bright and free from oxide, and in such a condition as to present the largest possible surface of metal for the immediate reaction of precipitation.

In collecting the precious metal precipitates any filter press may be used, but experience has demonstrated the superiority of the ordinary bottom filling press over any other.

In mills when cupellation is carried out atomized lead used for precipitation is of great benefit, as it reacts better, owing to its being well mixed with the gold and silver, and its state of fine division is also beneficial.

Atomizer.

The simplicity of construction and method of operation of the preferred form of "Atomizer" can be readily seen by referring to the accompanying photographs. Fig. 1 shows the atomizer ready for operation. Fig. 2 is a sectional view of the atomizer by means of which the following description will be easily understood:

The atomizer consists of an iron jacket divided into two compartments. The interior of this iron jacket is protected by a lining of fire bricks, having the necessary openings and connections.

The lower compartment is arranged, as shown in the illustration, so as to contain a superheating coil, surrounded and supported in position by fire bricks. In the lower compartment is also shown a combustion chamber, the upper compartment containing a superheating coil.

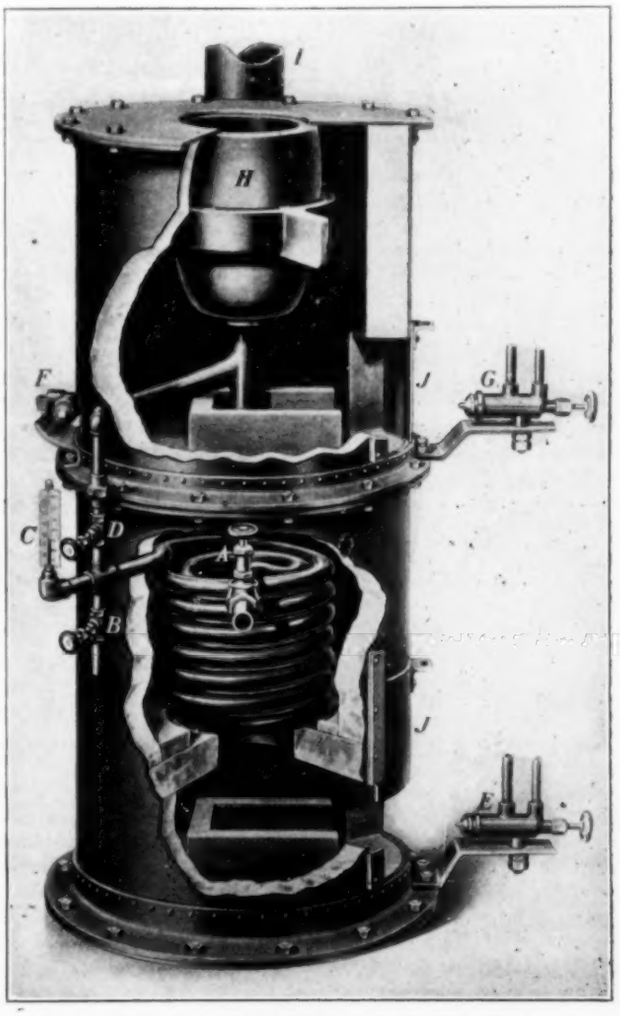


FIG. 2.—SECTIONAL VIEW OF ATOMIZER.

The superheating coil has a valve A placed so that the inlet of the steam or air to the superheating coil is regulated.

B is a drip valve, by means of which the steam or air passing through the coil is allowed to escape when it is not desired to pass the same through the atomizer.

C is a thermometer whereby the temperature of the steam or air being superheated is regulated.

D is a valve whereby the quantity of superheated steam or air required is admitted to the atomizer.

E is an oil burner to supply the necessary heat to the superheating chamber.

F is the metal atomizer.

G is an oil burner to supply the necessary heat to melt the metals undergoing treatment.

H is a crucible connected with the atomizer.

The upper chamber or compartment is simply a furnace to contain the crucible H.

Both the lower and upper compartments are connected with a chimney I to carry off the product of combustion from these compartments.

J J are doors whereby the temperature in either of these compartments may be regulated.

The atomizer is of special design, which allows of the molten metals first being formed into a film, before being atomized, thus insuring uniformity of product.

In operating the apparatus, heat is first applied by means of the oil burner to the upper compartment until the contents of the crucible commences to melt. Then steam or air is allowed to pass through the super-heating coil, and escape by the valve B. Heat is then applied to the lower compartment until the thermometer shows that the steam or air passing through the coil is at the correct temperature for atomizing. This temperature is maintained during the operation of atomizing.

Having thus reduced the metal to a molten state and with the steam or air heated to the necessary temperature, the metal is allowed to flow through the atomizer and then the drip valve B is closed and, by means of the valve D, the necessary amount of steam or air required to completely atomize the metal is allowed to pass through the atomizer into a collecting chamber.

When the superheated steam or air strikes the molten metal issuing from the atomizer, the metal is at once formed into a film and immediately atomized, leaving the atomizer in the form of a mist. This mist is deposited in the collecting chamber and on examination will be found to be in a pure unoxidized state of subdivision, its fineness being controlled at will by the operator.

The capacity of the apparatus shown is upwards of 100 lb. of atomized metal per hour.

The exclusive rights to the letters patents for the apparatus and the process of atomizing, and the process of using atomized lead and zinc in the precipitation of precious ore are owned by the Chemical Process Company, 170 Broadway, New York City.

The Use and Care of Crucibles in Foundry Practice.

Very little has been published on the manufacture and uses of graphite crucibles. A recent bulletin, issued by the Jonathan Bartley Crucible Company of Trenton, N. J., entitled "Crucibles and Other Graphite Productions for Foundry Use," is a notable exception. It is well written and well illustrated, and not only gives catalog data on the crucibles, retorts, etc., made by this company, but suggestive hints on the proper handling of graphite crucibles in foundry practice. The following information is taken from this bulletin:

Concerning the manufacture of graphite crucibles, much secrecy is preserved. Each manufacturer has his own secret formulas and methods. The margin of safety in computing formulas for different classes of work is very small, so that a slight deviation from a proper formula will prove disastrous. Different work requires different formulas; for example, a formula of composition which gives good results on steel will not give good results on brass; one suitable for nickel is not suitable for steel, etc. While analysis of the raw material is essential to produce a uniform crucible, an analysis of the crucible itself shows practically nothing beyond the limit of simple comparison. While chemical analysis is absolutely necessary, it is not everything, since a hundred different concoctions can be made that will apparently give the same analysis, but be widely different in physical properties.

Besides the careful selection of the raw materials in proper proportions, according to analysis, the manufacture of satisfactory crucibles also requires utmost care in the proper mixing of the raw materials, in the use of perfect molds and presses, and finally in proper and sufficient burning for the purpose of calcining the clay.

A laboratory test of the finished crucible amounts to nothing. The crucial test on which the crucible must stand or fall is made on the foundry floor, yet the life of a crucible in foundry practice depends to a large extent on the proper or improper use which is made of it by the founder.

The finished crucible has become hygroscopic as a result of the burning process. How much moisture it absorbs from the air will depend entirely on the atmosphere. In one instance a crucible, taken from a kiln of the Jonathan Bartley Crucible Company, was tested and found to contain less than one-quarter of one per cent. moisture, placed in a room where the temperature registered 90° F., and five hours later it contained two and seven-tenths per cent. moisture. It was then placed in the free atmosphere, and five hours later it contained four and one-half per cent. moisture. This illustrates the necessity of proper annealing of crucibles.

Every shop and foundry has its own method of **annealing** and this method depends very much on the conditions of the foundry. Thus, in one foundry few cores may be used and the oven can be almost entirely devoted to the annealing of crucibles; a second foundry might have an oven where cores are used extensively and the cores might be large. To place crucibles in an oven with a load of wet cores would be something in the order of putting a cat in the same cage with a canary bird. The crucibles would quickly absorb the moisture, and while it is true that a sufficient time and heat might eliminate danger to both, the practice is generally considered a dangerous one.

Another method of annealing which is quite popular among small foundries is to place a crucible on top of a furnace. This practice in most cases is all right provided the melter uses discretion. Crucibles, as a rule, when they arrive at a foundry contain from 4 to 5 per cent. moisture. If they are placed at once on top of a very hot furnace there is a chance of an immediate "scalp," and even if they should show no external damage, there might be caused an internal fracture which would in a few runs develop a leak, or what are commonly known as "pin holes."

The best device for annealing crucibles is a furnace built expressly for the purpose, using the waste heat from the foundry furnace on its way to the stack. These can be built so that the heat can be regulated and the temperature brought gradually from zero to or above 250° F. The average size crucible, say No. 60, should remain at this temperature at least 48 hours before going into the furnace. Larger sizes require more time.

The **tongs** require special notice. That they play a very important part in the life of a crucible is shown in the fact that in the tilting furnace, where the lifting in and out is eliminated, the crucible will run more than double the number of heats. Foundrymen as a rule are too careless in considering the importance of manipulating this end of their work.

When the crucible is under its melting heat it is plastic and readily adjusts itself to any irregular shape to fit the tongs. After a pour is made and the crucible cools off it will retain this irregular shape. In the second operation, unless the tongs are placed in the exact position, an unequal strain will occur, and if this is repeated time after time the crucible is weakened and naturally its life is shortened.

Different foundries have their own ideas as to the proper style of tongs best fitted for their work. But in any case utmost care is necessary to keep the tongs properly fitted to the crucibles. Reference is made to an exceptional case in which one pair of tongs handled three different sizes of crucibles in one shop. The loss in crucibles here would have paid for a dozen tongs in a short time.

In actual use there is nothing more detrimental to a graphite

crucible than the **fuel gases** that arise from impure fuel. Their effect permeates the entire wall.

Both coal and coke which contain sulphur to any great extent will produce this result, particularly if used in a wet or damp condition. Imperfect combustion with oil or gas will also act in the same way or even to a greater extent, due to the fact that in the coal or coke fire the crucible is almost completely submerged in the fuel and thus protected from extreme oxidizing conditions, so that the bad effect is obtained more at the top of the crucible, whereas in the oil or gas furnace imperfect combustion affects the entire surface.

These pernicious gases are always produced at low temperatures—usually in a coal or coke fire immediately after a fresh charge of fuel, and in the oil or gas furnace as a result of a slight manipulation of the air or fuel valves.

Great care should, therefore, be given to the fuel so as to have it as pure as possible, and perfectly dry, to charge it properly at the right time.

The life of a graphite crucible is oftimes greatly reduced by what is called in foundry parlance **soaking.** While this practice is more general among small founders, it is also observed to quite an extent in some of the larger plants.

At the beginning of the day's work the furnaces are fired, the charge made and the crucible placed. The molders begin their work; 9:30 a. m. finds the metal sufficiently melted for a "pour," but the molder is not ready, and it often occurs that two, three or four hours pass before the pour is made. All this time the crucible is simply living its life on wasted energy. Under a condition of this kind the ordinary life of a crucible can be divided by three.

The Latest Progress in Optical Pyrometry.

BY ARTHUR LENK, CH. E.

The optical pyrometer has never had a keen rival in the thermo-electric pyrometer in the correct measurement of high temperatures, as so far no material is known that, if used as an armor, can permanently withstand prolonged and direct exposure to intense heat. A further aggravation of such difficulties is to be found in the chemical activities of furnace products and gases, which in many cases make an adequate protection of the thermo-couple entirely impossible. In other numerous instances in practice the temperature measurements cannot be made with pyrometers which must be brought into contact with the material operated upon, on account of its inaccessibility through being in motion. In all such cases, and where the temperature of large areas of hot masses must be readily measured, pyrometers acting at a distance are required, and if it comes to the measurement of exceedingly high temperatures, up to 7000 deg. C., only such instruments can be employed.

FIG. 1.—VIEW OF SCIMATCO OPTICAL PYROMETER.

The most important point in favor of all optical pyrometers based on the well-known formula of Paschen and Wien is the fact that their indications are not based on "ascertained properties" of certain materials, such as the thermo-electric forces of the thermo-couple used with radiation pyrometers, but on a well-known infallible law of physics. As to the practical application of optical and radiation pyrometers, the former advantage of the radiation over the optical instrument through direct temperature readings on the Centigrade scale, in contrast with the readings from separate tables with the old optical type, has entirely disappeared with the recent improvement of the optical pyrometers through construction with direct-reading temperature scales.

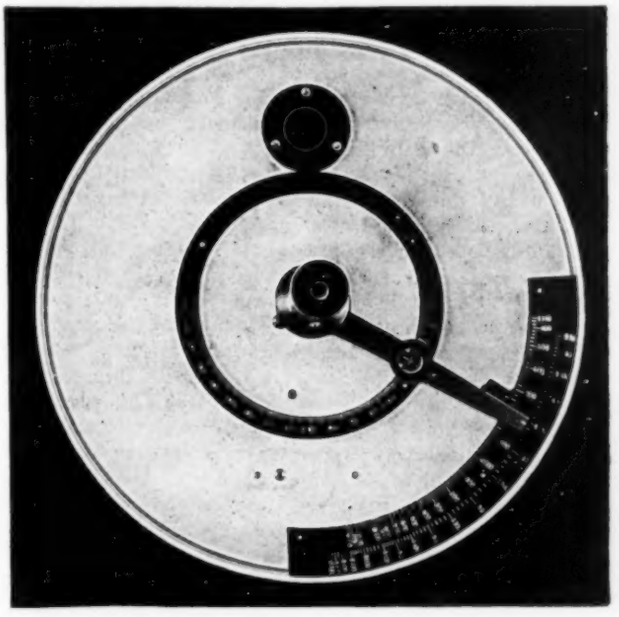


FIG. 2.—SCALE OF SCIMATCO OPTICAL PYROMETER.

The application of the Paschen and Wien formula in a commercial instrument has until now been confined to one instrument manufactured in Germany, and most detailed researches have been carried on for the last few years with the view of improving and perfecting the same. The many careful tests made in the laboratories of the Reichsanstalt, as well as at the furnaces of the most prominent German glassworks, were highly successful and resulted in the introduction of a perfect type on the American market.

Some of the important improvements of the Scimatco pyrometers, which have a range of from 650 deg. to 7000 deg. C., may be mentioned here.

The direct temperature scale of the Scimatco pyrometer is permanently fixed to the ocular. The improved construction of the Scimatco permits a most perfect adjustment of the direct temperature scale.

The direct temperature scale of the Scimatco has the further great advantage that the scale shows not only the temperature, but also the angle, on which the reading is based, permitting a reliable checking of same. The temperature readings with the Scimatco are, therefore, at all times reliable.

The detachable screw-cap of the steel tube protecting the Scimatco pyrometer is also an important advantage over other instruments similarly protected. The small electric lamp has to be renewed frequently, and it is of great importance in practice that the renewing of the electric lamp in the Scimatco can easily be done in a few seconds, without tools. The electric lamp of the Scimatco pyrometer is enclosed in the protective steel tube.

The large disk of the Scimatco pyrometer fully protects both eyes. This is important since experts have declared that it

is impossible to make reliable readings with only one eye open.

The temperature scale of the Scimatco is uniformly graduated in even numbers of degrees, thus facilitating readings and avoiding errors apt to occur with scales showing odd figures at irregular intervals.

The optical parts in the Scimatco pyrometer are inserted in the protective steel tube in such a way that they have no contact with the metal wall of the tube. If the instrument is used before large furnaces the pyrometers naturally become extremely hot. It is then of the utmost importance that the lenses and prisms which are partly cemented together be fully protected against heat and sudden changes of temperature. In the construction of the Scimatco pyrometer the necessary protection is not only rendered against the radiating heat by the surrounding steel tube, but even the conduction of heat is prevented, as the optical parts are separated by air from the heated metal walls of the instrument.

The screw-cap is absorbing the heat produced by the electric lamp and the small slit through which the light rays enter the pyrometer from the outside is constantly kept warm. There are, therefore, no expansions of metallic parts by heat with resulting changes of the size of the small slit through which the light enters the instruments. These would be a source of error because when a pyrometer is adjusted or standardized it is done in a laboratory, in a far lower temperature than the pyrometer finds in its practical applications. If the metal expands the slit gets smaller, and therefore the intensity of the light entering the instrument—the vital point on which the measurements are based—diminishes. The electric lamp in the Scimatco produces a constant temperature in the part bearing the slit and therefore successfully offsets the large changes in outside temperatures, while a considerable difference is caused in the dimension of the slit of other types through such outside influences.

The slit through which the light rays enter the Scimatco pyrometer is protected against dust and soot by a small glass slide of known optical properties, which have been most carefully investigated and fully approved by the Reichsanstalt. This device renders the instrument immune from dust and soot particles settling in the small slit and optical interferences caused thereby. If such protection is not employed, the use of such instruments in dusty atmospheres always gives rise to disturbances caused by dust and soot, a very disagreeable point, as it is difficult, sometimes impossible, to give the pyrometer a thorough cleaning. If the hair slit is covered by dust or soot the instrument is absolutely useless.

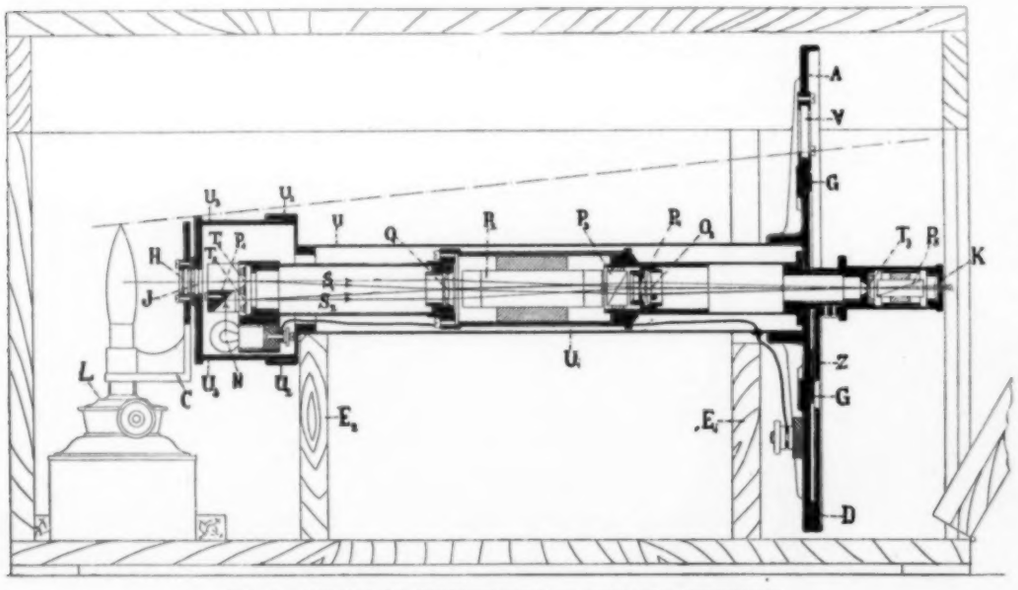


FIG. 3.—DETAILS OF CONSTRUCTION OF SCIMATCO PYROMETER.

Another very important improvement is the way in which the electric current is supplied to the small electric lamp of the Scimatco pyrometer. It has been fully ascertained that a great danger of fluctuations in the electric current exists, if the wires are attached to small leads to the contact of the lamp. First of all, the small leads cannot be properly and readily cleaned, and furthermore the wires are always more or less liable to tension, thus making the contacts unsafe and unsatisfactory. The system of supplying the electric current to the electric lamp of the Scimatco pyrometer is perfected in every way. The very large leads are fixed on the rear side of the temperature scale, where they can always be readily inspected, cleaned and tightened, and from there the wires are conducted to the lamp within the steel tube, fully protected against all disturbing influences.

The Scimatco pyrometer is constructed throughout in a very solid manner. The inner parts are fixed in such a way to the instrument that the whole arrangement acts like a spring whenever the apparatus suffers a shock through dropping, etc. Breakage of the Scimatco pyrometer is therefore hardly possible under the most straining circumstances.

Everybody familiar with the handling of optical pyrometers and realizing the many important improvements outlined above will readily see that the Scimatco instrument is the result of many years' research. It has been placed on the market in this country by the Scientific Materials Company, Pittsburgh, Pa.

Purity a Requisite for Silver Electroplating Anodes.*

By J. L. R. Brown.

The best way to show that "purity" is a requisite in a silver anode, is to point out some of the bad results arising from the use of impure silver, all of which bad results might have been avoided by the use of silver prepared for the purpose of plating and fully guaranteed. Let us review these bad results.

I am sure all electroplaters (especially in the old days) have had the disagreeable experience of contending with the presence on their silver anode surfaces of a black film attached so lightly that unless carefully handled it would float off into the solution, but which no amount of work, such as altering resistances, changing solution conditions or the many things one will do on occasions of this sort, could prevent. The only thing to do is to carefully remove them, wash thoroughly and replace, and continue this operation until the anodes are consumed.

Not all of the anodes would work this way, and sometimes not all of the surface of the anode would turn black. It would form in streaks alternating with clear portions, and under these conditions, the free cyanide would act correctly on parts of the surface and very little if at all, on others; at times eating through the anodes in the clean portions causing the black, where surrounded by the clean, to become detached and

*A paper read at the third annual banquet of the National Electroplaters' Association on February 10 ni New York City.

sink to the bottom of the tank. The ideal way for an anode to go into solution is, of course, to dissolve evenly all over the surface and to disappear from the edges, preserving the contour in the form of a regular curve.

The appearance of the silver anode before using is no guide in determining whether the anode will work white or not, as the surface of an anode which will work black is as clean and white before going into the solution as any other which is perfectly pure. In fact, there is little doubt but that the silver entering into a black anode is 0.999 fine, which it is necessary for it to be in order that it may be sold as "commercial silver." The term 0.999 fine, however, simply means that in every thousand ounces of bullion, there are 999 oz. of pure silver and 1 oz. of alloy. When this ounce of alloy is all, or partly composed of insoluble matter, the action of the free cyanide in the solution, dissolves out the silver in the anode, and leaves the insoluble matter in the shape of a black film on the surface. This will accumulate so rapidly when the anode is bad as to necessitate removal and cleaning as often as once a day.

I realize that silver anodes will turn black, when the silver from which the anode was made is absolutely all right. This may arise from a number of conditions, such as the action of sulphur in a bright solution, too high a current density for the work to be plated, or too much or a poor quality of cyanide. When the discoloration arises from faulty conditions in the plating room, it is of a different shade and easily distinguished, and a good plater can easily discover the cause and apply the remedy. The only remedy I ever heard for a black anode where the silver was bad, was to replace the bad ones with good ones.

Among the bad results arising from the use of impure silver, may be mentioned the following:

The increasing amount of resistance caused by this film lessens the rate of silver deposition and in the absence of scales at the tubs, the customer will get lighter plate than he pays for or than the plater intends to give him, or else, if the plater is careful to get the weight of silver on the goods, it takes him so much longer that the output of his department is lowered.

Further, the presence of this film retards the dissolving action of the free cyanide in the solution, and in a slight degree, the anode becomes neutral so that the silver is plated out of the solution at a greater rate than the free cyanide can replace it. I believe the first impulse of a plater, under this condition, is to increase the amount of free cyanide so as to take up the silver faster. This, however, is only a palliative and accomplishes no lasting good, as the solution is soon out of balance, and requires refining, and replacing by a new one.

One member of this association has told me of his solutions which have been in use nearly five years, and the quality of the anodes has been so uniformly good that in all that time he has not been obliged to supply any silver salts.

When the periodic cleaning of the black anode is neglected, the film becomes thick and floats off into the solution and not only contaminates it, but the dirt is carried over on the work. This makes it necessary to remove the goods and clean them with the danger of getting grease on the surface, and interfering with the smoothness of the plate. If the dirt gets on the goods and is not noticed, it is plated over and the plate blisters and tears out in burnishing or buffing, and requires stripping, refinishing and replating. That is what happens when you do not remove the anodes and clean them. Now if you do take them out, the action through the solution in lifting them, rinses the dirt off into the bath, and you get it anyway.

In the old days, when it was necessary to use any kind of silver which could be obtained, I understand it was the practice of some platers to hang a cloth curtain in front of the anodes to keep the dirt from floating over to the work, and while it would accomplish this purpose, it was still necessary to wash the anodes, and in addition the curtains needed washing, and the removal of the anodes and the curtains agitated the solution, and rinsed off the dirt. Of course, these troubles would not have arisen had the entire anode been soluble in the solution, as all of this residue must have been present in the anode when it was put in the solution, and the action of the cyanide solution taking out the pure silver in the plate, left the black residue, because the cyanide was not a solvent of it.

I know of electroplaters who bought the regular 1000-oz. commercial bars, brought them to a low heat in a forge fire, put them under a drop hammer and flattened them to a point where they would enter their rolls and others who melted these bars, pouring the silver into flat shallow moulds thin enough to enter their rolls, and from this point, with suitable annealing rolled them to the right gauge and width, and cut to suitable lengths for anodes. Others bought what are known as Government assay bars, which are cast in a mould turning out a bar thin enough to be rolled without remelting. These used these bars feeling confident that silver bearing the stamp of the United States was the very best which could be obtained.

As a matter of fact, the government stamp on a bar of silver simply means that the melt from which that bar was poured was assayed and the fineness stamped on the bar is the result of such assaying. These bars may or may not contain insoluble matter, and are, therefore, we think, not the best silver for the purpose.

The point of view of the mints and the assay office is entirely a different one from our own. Our object is to establish a standard for anode silver and to uniformly maintain that quality; while the Government in refining the silver simply uses it as a carrier for the gold.

The proportion of silver in the dore which the Government requires in the process of gold refining used by them is so high that the immense amount of gold turned out by them requires a far greater quantity of silver, and this silver is furnished the Government absolutely free, and they are paid to use it.

One takes in gold and silver bars to be refined, they return to him the silver which they do not want, pay him for the gold which they do want and charge him for the refining. It is readily seen the only interest the Government has in the silver is to use it as an aid to their gold refining and to so carefully part the bars as to leave the silver free from gold and be able to turn it out 0.999 fine.

There are two branches of industry using silver which should have the purest which can be obtained to get the best results, one the electroplaters, and the other the manufacturers of silver nitrate; while at present the electroplaters use the larger amount, yet as a result of the recent popularity of the moving picture show, of the almost universal use of the picture post card (I speak of the sensitized kind) and of the coming among us in such large numbers of the amateur photographer, the consumption of silver in the manufacture of nitrate is much larger than is generally known.

Notwithstanding the fact that silver plating was being done commercially over 60 years ago, there was no concern manufacturing and marketing silver anodes as a commercial article until the author's firm originated the idea, and so far as I know, made the first silver anodes for sale as such at their plant which was then located in Chicago.

In talking with a plater, who at the time was engaged in washing off his anodes and skimming his solution, we asked if it would not be a good thing if an anode could be made which would always work clean and white and guaranteed by the manufacturer to be uniform from lot to lot. From the encouragement received at this time, we commenced experimenting and after numberless discouragements and expensive experiment we succeeded in 1903 after we had moved our plant to Bridgeport, Conn. Since then, we have made and sold tons of anodes, every ounce of which is thoroughly guaranteed.

The silver requires very careful selection to start with and careful handling all through the process. There is very little silver ore which is mined and smelted as such, the value per ton not commercially warranting the operation. There are exceptions to this, of course, the most noted one being the mines

located in the Cobalt district. In many of these, the total cost for mining, smelting, and marketing the silver, is from 12 to 25 cents per ounce. The present level of the silver market (at around 60 cents per ounce) permits the operation of these mines at quite a fair profit.

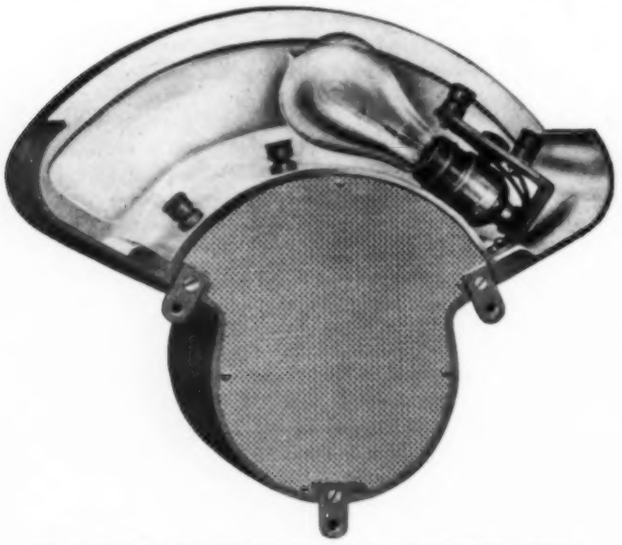
The bulk of the silver used by our firm, I think I may safely say nearly 85 per cent, is produced as the result of smelting lead, copper and others ores, where the silver is left behind in the slimes with any impurities. Naturally as the ores vary a great deal in impurities, depending upon the locality from which they are mined, so also the impurities accompanying the silver will vary, and while some silver is suitable for anode purposes, others, owing to the excessive cost of suitably refining, are prohibitive, while others in turn cannot be used at all. The success depends on being able to select the right kinds, and rejecting the wrong kinds, in fact in knowing how.

Handy & Harman,
New York City.

Illuminated-Dial Pyrometer

The Brown Instrument Company of Philadelphia are adding to their very complete line of electrical pyrometers, a new illuminated dial indicator, with exceptionally long scale. Frequently an instrument is desired which can be easily read by the operator of the furnace or kiln, at some distance, and the new Brown pyrometer with illuminated dial is particularly desirable for just such conditions.

This new instrument with extra long scale 12 in., can be furnished with or without illuminated dial as desired. Where the



BACK OF ELECTRIC PYROMETER WITH ILLUMINATED SCALE.

illumination is sufficient, the instrument does not require an illuminated dial, but the long scale affords very open graduations and accurate readings.

In many plants, however, such as dark shops for hardening and heat treating steel, and where work is carried on day and night, the illuminated-dial instrument is preferable. The scale is drawn on opaque glass, and a 16-candle power lamp, placed behind the scale, makes the graduations and the pointer stand out clearly at a distance. The operator using the instrument appreciates the advantages of such an instrument, and better results are secured through the ability to obtain more accurate readings of the temperature.

The Iron and Steel Market.

The active steel market which was expected this year has not materialized. February has not proved better than January, and while both months represented better conditions than prevailed, on an average, during the second half of 1911, there has not been the return to the active conditions of November and the fore part of December that was expected.

The mills have been operating in the past two months chiefly on the accumulation of business in the closing months of last year, but there has really been a fair amount of business done, chiefly by way of specifications being entered on contracts then placed. Of strictly new buying there has been relatively little.

Mill operations have steadily increased in the past two months, from an average of 75 per cent to full capacity, maintained quite steadily during the fourth quarter of 1911, to about 85 per cent at the present time. Much of the additional capacity was nominally put in operation in January, but on account of extremely severe weather outputs did not correspondingly increase, but rather were somewhat smaller than in December. Toward the close of February actual outputs became decidedly more satisfactory.

The mill activity has not been equally distributed by any means. The heaviest pressure has been in the Steel Corporation mills in the Pittsburgh district, partly on material for the large number of cars placed with the car shops in October, November and December, and partly on export business, with which the Steel Corporation continues to be well supplied. Independent producers in the Central West have been operating fairly well, while the Eastern mills, as a rule, have had very light operation, generally ranging from 50 to 65 per cent of capacity.

The merchant furnaces have not increased their production as compared with last year, thus being in a distinctly different position from the steel interests. It is now possible to estimate with considerable accuracy the extent and importance of the buying movement which occurred in November and December. It has been made evident that it did not represent any actual increase in the country's rate of ultimate consumption of iron and steel products. Two influences were present. In the first place, prices all along the line had dropped to the lowest point since 1899, and in a few cases to levels lower even than the historic low levels of 1897 and 1898. In the second place, a number of railroads made fairly large purchases of cars and smaller purchases of other material. While in this action they may have been moved partly by the need of equipment and partly also by the very low prices prevailing, there is no doubt that there was a definite understanding among some important men of affairs that it would be of general advantage if the market were given a stimulus by the placing of some large orders. Through these two influences a buying movement was started in other quarters by jobbers and manufacturing consumers. Thus the active conditions of November and December did not represent a definite increase in the rate of consumption, but a burst of buying which was necessarily temporary in its character.

There is reason to expect an improvement in demand in March and April, but even that would not prove that the present year was marked for one of continued high activity, for the reason that consumptive requirements along certain lines have been accumulating on account of the extremely unfavorable weather, retarding or entirely stopping outside operations. Many large jobs have fallen a month or more behind their schedules, and with spring weather efforts will naturally be made to catch up. Thus a sudden increase in demand may occur in the near future, but it is also quite possible that provision has already been made for this increase, and that in anticipation the mills have made more material, and the buyers have taken more material, than was currently needed in the past two months.

Nominal prices for finished steel products have suffered to the extent of declines of about a dollar a ton in bars, plates, shapes and sheets. These are real declines, to the extent that material could not have been bought at present prices thirty days ago, but they are not declines from the average prices prevailing on material recently shipped, for shipments have

been almost entirely upon purchases made before the advances in December, and the market remains quotably higher on several prices, and in no cases lower, than the level prevailing at the low point in November. Plates, bars and shapes, however, are now quotable at 1.10c., Pittsburgh, against 1.15c. at the close of December, while sheets are quotable at 1.85c. against 1.90c.

Pig Iron.

The pig iron market has been dull throughout February, but in this respect no change was shown from January. The merchant furnaces have not increased their production as compared with January or December, but are shipping their product as fast as made now as was the case in those months. During 1911 stocks at merchant furnaces decreased and the statistical situation is good. It would require only a slight increase in actual consumption, probably to send prices up fractionally, but no large advance is possible on account of the many furnaces idle. With each advance of, say, 50 cents a ton, a considerable number of furnaces would resume operations. The merchant furnaces are much interested in ore price prospects for the coming season. Most of them have considerable stocks, but of course will have to buy for the season. For last season several central western furnaces did not buy at all. There will be a reduction in Lake Superior ore prices for the season, as compared with last year. The rail rate from the Mesabi range to Lake Superior was recently reduced 20 cents, which saving will be given to consumers of Mesabi ore, while the dock charges at Lake Erie ports are likely to be decreased by about 10 cents through an expected order of the Interstate Commerce Commission, which would reduce the cost of delivery of ore from all the ranges. Apart from these reductions Lake Superior ore on Lake Erie dock will be lower, probably by from 50 to 75 cents on Mesabi and 25 to 50 cents on old range. In the past two years the eastern furnaces have been using imported ore. With the advance in ocean freights and in ore prices at foreign mines, together with the expected reduction in Lake Superior ores, there has been a chance for the Lake Superior ore interests to regain a portion of the eastern trade. When, however, they approached the eastern furnaces with promise of a reduction, the eastern furnaces reflected that this would give the Buffalo district cheaper ore also, and thus place the eastern furnaces at a disadvantage, when competing with the Buffalo furnaces in New England territory. The freight on ore from Lake Erie to most of the eastern furnaces is $1.45, whereas last year the eastern furnaces obtained foreign ore costing less per unit than the cost at Buffalo, plus the $1.45 freight. The next move of the Lake Superior ore interests was to appear before the Finance Committee of the Senate, in the tariff hearings, and ask that the ore duty be made 50 cents. It was 75 cents in the McKinley law of 1890, the Wilson-Gorman law of 1894 and the Dingley law of 1897, being reduced to 15 cents by the Payne-Aldrich law of 1909. Since the independence of Cuba was established that country has had a preferential of 20 per cent, making the present duty on Cuban ores 12 cents. The Underwood bill which recently passed the House placed ore on the free list. Naturally, there is no likelihood of Congress at this time making an advance in the duty, and particularly an advance to a rate higher than has prevailed for nearly 18 years. Pig iron is quotable as follows: Southern iron, $10; Birmingham, for prompt, with $10.25 to $10.50 asked for second quarter; No. 2X foundry, Philadelphia, $14.75 to $15; No. 2 foundry, f. o. b., Buffalo furnaces, $13.75; No. 2 foundry, f. o. b. Chicago furnaces, $14; f. o. b. valley furnaces (90 cents higer delivered Pittsburgh); No. 2 foundry, $13 to $13.25; gray forge, $12.50 to $12.75; malleable, $13 to $13.25; basic, $12.25 to $12.50; Bessemer, $14 to $14.25.

Steel.

Consumers are well covered by contracts for billets and sheet bars, but in many cases are not receiving full deliveries, and there has been considerable demand for small lots for prompt shipment. These would perhaps command a slight premium over forward deliveries. The market stands approximately as follows: F. o. b. maker's mill, Pittsburgh; Bessemer and open-hearth billets, $20 to $20.50; Bessemer sheet bars, $21; open-hearth sheet bars, $20.50 to $21; f. o. b. maker's mill, Youngstown; Bessemer billets, $19.50 to $20; Bessemer sheet bars, $20 to $20.50. Wire rods are nominally $25, Pittsburgh.

Finished Material.

Current prices, f. o. b. Pittsburgh, unless otherwise stated, are as follows, and are fairly well maintained:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mills, except Colorado.

Plates, tank quality, 1.10 cents; Chicago, 1.25 cents.

Shapes, 1.10 cents.

Steel bars, 1.10 cents; 1.25 cents, Chicago.

Iron bars, 1.25 cents, Pittsburgh; 1.25 cents, Philadelphia; 1.15 cent, Chicago.

Wire nails, base, $1.60; plain wire, 1.40 cents, base.

Black sheets, 28-gage, 1.85 to 1.90 cents; galvanized, 2.85 to 2.90 cents; painted corrugated roofing, $1.33 to $1.35 per square; galvanized, $2.47 to $2.50 per square.

The Non-Ferrous Metal Market

The February metal market was characterized principally by a marked reduction in lead, made by the American Smelting & Refining Company. Business in the tin and copper markets was only fair, but in lead and spelter there was considerable activity.

Copper.—The feature of this market was the nominal quotation and light demand for Lake copper. Electrolytic was in considerable demand both for European and domestic consumption. With the reduced prices for electrolytic, the business done in Lake was insignificant. The latest quotations are slightly higher than those which prevailed early in the month. Electrolytic, 14.08 cents; Lake, 14¼ cents.

Lead.—All the selling agencies were forced to meet a marked reduction made by the American Smelting & Refining Company, and the market has been active at 4 cents, New York, and 3.87½ cents, St. Louis.

Tin.—This metal was available in quantities to suit, and the premium for spot metal was removed. February tin was quoted at 43⅛ cents.

Spelter.—Large demands for future delivery have been met by the smelters, and there has been a premium demanded in some cases for spot metal. The market for current delivery was active at 6.50 cents, New York, and 6.35 cents, St. Louis.

Other Metals.—Business in aluminium and antimony has been quiet, but quicksilver has enjoyed an advance in price, with steady demand. The latest quotations are: Aluminium, 19 to 19½ cents, New York, for No. 1 ingots. Antimony, 6¾ to 7⅜ cents for various brands. Quicksilver, $44 per flask of 75 lb., New York, and $43.50, San Francisco.

Personal.

Messrs. Philip Argall & Sons, Denver, Col., have been appointed consulting engineers in charge of the properties owned by the estate of the late David H. Moffat, of Denver, Col.

Mr. Gelasio Caetani, of San Francisco, has been at Telluride, Col., on professional business at the large mills there. Mr. Caetani recently has entered into partnership with Albert Burch and Oscar H. Hershey under the firm name of Burch, Caetani & Hershey, consulting engineers in mining, metallurgy and mining geology. Their offices will be in the Crocker Building, San Francisco.

Mr. George E. Collins has returned to Denver from London.

Mr. George A. Guess has been appointed professor of metallurgy in the Toronto University, Canada.

Mr. William G. Haldane, assistant professor of metallurgy, Colorado School of Mines, was married on February 28 in Denver to Miss Lorena V. Beaver.

Mr. Carl Hering, the distinguished consulting electrical engineer of Philadelphia, has received from the University of Pennsylvania the honorary degree of Doctor of Science. Dr. Hering is a past president of the American Institute of Electrical Engineers and of the American Electrochemical Society.

Mr. Richard A. Parker has returned to Denver from a professional trip into Arizona.

Mr. W. C. Patterson, of London, patentee of Patterson agitation tank, is now located at Denver, Col., where he has a testing plant at the works of the McFarland Manufacturing Company, 3309 Blake Street, Denver, Colo.

Mr. Frank H. Probert has returned to Arizona following a trip to New York, where he was in consultation with the officers of the Ray Central and Proprietary Mines companies.

Mr. Frank E. Shepard, of the Denver Engineering Works Company, addressed the mining men of Telluride, Col., on Feb. 23. His subject was "Modern Mill Design."

Mr. F. M. Simonds, of New York, was in Denver recently on his way to examine the Rawley mine, near Salida, Col. While in Denver he investigated late improvements in ore dressing.

Mr. Norval J. Welsh, of San Antonio, Tex., has been appointed general manager of the Mexican Mines Development Company, operating in the Santa Eulalia district, Chihuahua, Mexico. His headquarters will be at 229 Avenida Independencia, Chihuahua, Mexico.

Obituary

Col. A. G. Brownlee, of Idaho Springs, Col., died at his home at the Stanley Mines, near Idaho Springs, on January 16, 1911. Col. Brownlee was a prominent operator of Colorado mines and mills, having been connected principally with the Japan-Flora property in the Telluride district, and with the Stanley at Idaho Springs. He was interested in the development of metallurgical processes, especially such as gave promise of treating successfully mixed sulphide ores, and in the last three years had given time and money to the application of dry chlorination. He was a director and vice-president of the American Mining Congress, and a past president of the Colorado Scientific Society.

NOTES.

Capacity of the Sulphur Mine of the Union Sulphur Co.—In Mr. Herman Frasch's paper published in our February issue there appears on page 78, second column, the following paragraph: "In order to demonstrate what the mine could do with six wells, we pumped that number simultaneously for *six months,* producing 122,000 tons of sulphur, which is more than the consumption of the whole world for that period of time. The words "six months" are a typographical error and should read *"two months."*

The occurrence of lorandite in the Rambler mine, Wyoming, has been established by Mr. Austin F. Rogers, who publishes his work in the February, 1912, issue of the *American Journal of Science.* The Rambler mine has been remarkable for its great variety of copper minerals, together with minerals of the platinum group. The determination of the occurrence of lorandite adds another rare mineral to the long list. This is a metasulpharsenite of thallium, $TlAsS_2$. The only other known occurrence of lorandite is at Allchar, in Macedonia.

The Dwight-Lloyd sinter-roasting furnace has been adopted in connection with the manufacture of sulphuric acid. The furnace delivers a continuous volume of uniform SO_2 content, averaging about 4 to 6 per cent SO_2. A large plant has been erected at Stolberg, Germany, in which these furnaces will be used in connection with the contact process. The Huntington-Heberlein roasting process also is used in acid manufacture, but the conditions are not so favorable as with the Dwight-Lloyd apparatus.

The consumption of sulphuric acid in European countries has increased in the past few years owing to the increased development of the superphosphate industry, and to the increasing number of coke ovens which manufacture by-products.

Calendars.—Quite a number of very artistic calendars for 1912 have been received. That of C. W. Leavitt & Company has an interesting picture of a 15-ton Girod electric steel furnace. The calendar of the Roessler & Hasslacher Chemical Company contains the international table of atomic weights for 1912, besides a fine picture by Coomans, while John MacWhirter's "Gates of the Forest" is found on the very artistic calendar of the Ashland Fire Brick Company. The calendar of the Goldschmidt Thermit Company has a map of the world and interesting photographs of thermit welds; the calendar of the Denver & Rio Grande Railroad shows a series of striking and stunning views of scenes along this "scenic railway of the world."

The Wellcome Photographic Exposure Record and Diary for 1912, issued by Burroughs, Wellcome & Company, of New York and London (price 50 cents), is as neat and interesting as any former editions of this annual. The information on photographic subjects which it contains within a small space is really remarkable. Attention is confined by no means to one plate or one method; on the contrary, brief and explicit instructions and explanations are given on such diverse topics as factorial and time development, machine, tank or stand development, the oil pigment process, contact printing by artificial light, color photography, development in warm weather and in tropical countries, printing and toning, color effects by staining, the actinic value of light in various latitudes, etc. The Wellcome exposure calculator is again added.

The McGraw-Hill Book Company announces the completion of arrangements for the sales agency of the technical books of the *Railway Age Gazette, American Engineer* and *Signal Engineer.* This makes the McGraw-Hill Book Company, publishers and distributors of books for the three journals, but it does not affect the publication of the papers themselves in any way. With the addition of these three papers in the railroad field, the McGraw-Hill Book Company now publishes books for the twelve leading American engineering papers in the fields of civil, mechanical and electrical engineering, electric and steam railroading, machine tools, mining, metallurgy, chemical engineering, etc. The papers now represented are *American Engineer, American Machinist, Coal Age, Electric Railway Journal, Electrical World, Engineering News, Engineering Record, Engineering and Mining Journal, Metallurgical and Chemical Engineering, Power, Railway Age Gazette, and Signal Engineer.*

Digest of Electrochemical U. S. Patents.

Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

Ore Treatment (*Continued*).

531,169, Dec. 18, 1894, Victor Engelhardt, of Vienna, Austria, assignor to Siemens & Halske, of Berlin, Germany.

Antimony and Arsenic; also gold, platinum, tellurium, molybdenum and tin, from sulphide ores. The ground ore is dissolved, with agitation and at a temperature of 212° F., in sodium

sulphydrate, with evolution of hydrogen sulphide. The solution is filtered and electrolyzed in a cell having a diaphragm of sheet asbestos boiled in gelatine, the anodes being of carbon or platinum, in a closed compartment, the cathode of copper or antimony. The antimony or arsenic are electrodeposited, with regeneration of the sodium sulphydrate. If the ores contain gold, silver, copper, mercury, bismuth, zinc, cobalt or nickel, the anode electrolyte may be an alkali-metal chloride, and the evolved chlorine is then removed and used to dissolve these metals from the residue undissolved by the sulphydrate. The chloride solution is filtered and the metals are precipitated by hydrogen sulphide from the first operation.

537,423, April 9, 1895, F. H. Long and D. C. Skaden, of Chicago, Ill., Long, assignor to A. G. Long.

Precious metals. Dissolves and deposits the metals electrically in a horizontal tumbling drum. The inside of the drum is lined with carbon plates having ore-lifting wings, serving as cathodes. Supported axially within and rotating with the drum is a perforated iron tube, serving as cathode, and surrounded by a jacket of asbestos cloth. Within and fitting this tube is a fixed screw-conveyor. In use, the ore pulp, mixed with some chlorine-generating substance is introduced into the drum, which is set in rotation. The metals are dissolved by the evolved chlorine, and are redeposited in the perforated iron tube, from which they are scraped off and carried out by the conveyor, one end of the tube opening through one head of the drum. A fixed gas vent pipe extends through the other head.

538,522, April 30, 1895, Edward D. Kendall, of New York City.

Gold and Silver. Precipitates from cyanide solutions by amalgamated zinc granules, turnings, filings or shavings. The residue zinc is removed from the precipitate by covering it with granulated hard carbon and adding dilute sulphuric acid.

543,546, July 30, 1895. Edwin J. Fraser, of San Francisco, Cal.

Gold and other precious metals. Precipitates from acidulated solutions by passing them longitudinally through a cylindrical vessel containing the elements of a voltaic pile, to wit, perforated zinc plates covered on both surfaces with cloth, alternating with bodies of pulverized charcoal.The end electrodes are preferably connected by a wire.

543,673, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Pourtales, of Silesia, Germany.

Relates to the cyanid process as applied to gold ores, the invention consisting in using an electric current to increase the activity of the cyanid solution before it is employed for lixiviation. It is stated that during electrolysis of a weak solution of potassium or other soluble cyanid, between an anode of carbon and a cathode of copper or iron, the electrodes preferably separated by a diaphragm, the solution surrounding the anode acquires greatly increased solvent powers of gold, with "no corresponding increase, but rather a decrease in the facility with which the various cyanid salts of the base metals are formed." The solution surrounding the cathode may consist of potassium cyanid or sodium sulfate. The effect at the anode is attributed to the formation of a certain proportion of cyanate, the oxygen of which is thought to be available for the solution of gold in accordance with Elsner's equation. Satisfactory results are secured when the proportion of cyanate to cyanid in the treated solution lies between 25 and 100 per cent. The data given in the patent would indicate an enormous increase of the activity of the liquid, as compared with pure cyanid. Following the lixiviation, any larger particles of gold may be recovered by amalgamation, preferably aided by the electric current, during which process some of the dissolved gold may be precipitated upon the mercury. The gold is then completely precipitated by electrolysis, and the solution regenerated. The advantages alleged for the process, in addition to the saving of time in lixiviation, are an increased stability of the solvent, the facility of regeneration of the liquid, and the producton of somewhat purer solutions of gold.

543,674, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Paurtales, of Silesia, Germany.

Relates to the solution of gold by the conjoint action of cyanid and cyanate of potassium, as in the preceding patent (543,673), the distinctive point of the invention being that the desired percentage of cyanate is added directly, as such, to the solution of cyanid. The proportion of cyanate, and the advantages attending its use, are as in the preceding patent.

543,675, July 30, 1895, Middleton Crawford, of Colorado Springs, Colorado, Assignor of three-fourths to Thomas C. Parrish and Duncan Chisholm, of same place, and James Pourtales, of Silesia, Germany.

An apparatus for treating gold ores by a solvent liquid, presumably the combined cyanid and cyanate of patent 543,673. The apparatus comprises two or more rotary barrels of wood, with provision for introducing the solvent through their trunnions. Following the treatment with the solvent liquid the contents of the barrels are discharged alternately upon a vibratory amalgamator for the recovery of any coarse gold. The tailings from the amalgamator are separated from the solvent liquid by filtration, and the latter is passed through an electrolytic cell in which the precious metal is precipitated. The solvent, thus regenerated, flows into a storage tank whence it is returned to the solution barrels.

BOOK REVIEWS.

A Manual of Fire Assaying. By **C. N. Fulton,** Professor of Metallurgy, Case School of Applied Science. Second Edition, 6 x 9¼ in. (15 x 23½ cm.), 219 pages, 60 illustrations, price $2.00, net. New York. McGraw-Hill Book Company.

In noticing the first edition of this book, we commended it as the first truly scientific and satisfactory work on assaying, treating the subject from the modern standpoint of exact science, in place of the old rule-of-thumb, cook-stove-recipe style. It should suffice now to say that the second edition, entirely rewritten and enlarged, is better than the first. *Nuf sed.*

* * *

Journal of the Municipal School of Technology, Manchester.—A record of investigations undertaken by members of the teaching staff and students of the school during 1910. Vol. IV, 253 pages.

Among the papers published in this volume there are the following: On the condition of indigo white in aqueous solution, Edmund Knecht; gravimetric methods for the determination of nickel in nickel steel, E. L. Rhead; apparatus for the study of gas-caps on the flame of an ordinary safety-lamp, G. H. Winstanley; crysallized caustic soda, F. S. Sinnatt; electrical theory of dyeing, W. W. Haldane Gee and W. Harrison; researches on bleaching powder, R. L. Taylor.

* * *

Technology and Industrial Efficiency. Papers presented at the semi-centennial of the Massachusetts Institute of Technology; 6 x 9 in., 486 pages; price, $3. New York: McGraw-Hill Book Company.

A collection of 59 papers, all written by graduates of the Institute, and presented at a technological congress held in Boston, April 10, 1911, to celebrate the fiftieth anniversary of the Institute.

There is an address by President Maclaurin on "The Factors of the Institute's Success," followed by seventeen papers on "Scientific Investigation and Control of Industrial Processes," thirteen on "Technological Education in Its Relations to Industrial Development," nine on "Administration and Management," fifteen on "Recent Industrial Development," nine on "Public Health and Sanitation," and four on "Architecture."

The range of subjects and high standing of many of the writers make this a very attractive book for those desiring a general review of the present relations of applied science to industry, commerce and public utilities.

* * *

The Heat Treatment of Tool Steel. By **Henry Brearley.** 5½ by 8¾ in., 160 pages, 73 illustrations. Price, 10/6 net ($3.50 net in New York). London and New York: Longmans, Green & Company.

The author has been twenty-seven years in the employ of Thomas Firth & Sons, Sheffield, and is now in charge of their research laboratory. His thirteen chapters deal with structure and classification, fractures and external appearance, forging, annealing, physical changes, hardening of steel, tempering, hardening typical tools, defective tools, hardening plant, pyrometers, case-hardening, alloy steels.

On inspecting the work we are impressed by the very clear, practical manner in which it is written. The absence of speculative theory and the directness of the information. It is admirably adapted to teaching students and beginners the principles and practice of the subject, and for explaining to tool-steel workers the real reasons for the manipulations practised in their trade.

* * *

The Metallurgy of Iron and Steel. By Bradley Stoughton, Ph.B. Second edition, thoroughly revised and entirely reset, 6 by 9 in. (15 x 23 cm.), 537 pages, 370 illustrations. Price, $3.00. New York: McGraw-Hill Book Co.

Mr. Stoughton gave a pretty satisfactory book in his first edition, and this second edition is still better. There is a new chapter on Metallurgical Fuels (well condensed), a better statement of the theory of the Bessemer process, and a new section on special open-hearth and duplex processes (brought well up to date). On page 473, the statement that 1 caloric per cubic meter = 8.9 B.t.u. per cubic foot is upside down; similar looseness in giving numerical date is observable in other places, such as 7,110,000 for 711 on page 33. These are, however, very minor defects in a book which as a whole, particularly on the descriptive side, is excellent and reliable. It is a particularly useful book for students' use, when supplemented by lectures upon the finer points of the subject. The publishers are to be commended for the fine typography and the low price.

* * *

Grundzüge des Eisenhüttenwesens. Von Dr. Ing. **Th. Geilenkirchen.** I Band. Allgemeine Eisenhüttenkunde. 249 pages, 66 cuts, 8 plates. Bound in cloth. Price, 8 marks ($2.50 in New York). Berlin: Julius Springer.

The complete work is to consist of three volumes, the whole intended as a strictly scientific exposition of the principles of iron metallurgy, omitting general descriptions and merely historical material. The three sections of this first volume treat respectively of industrial kinds of iron from the standpoint of the consumer, a compilation of the occurrences of iron ores in nature, together with a short review of the theoretical possibility of producing commercial iron from such raw materials, and finally the ways and means of producing and using heat as the agent of metallurgical operations.

The whole scheme is grandiose, but rather vague; the detail, as exemplified in this first volume, is hardly satisfying to the reviewer. It takes industry to compile satisfactorily, to fill out a larger scheme, but genius to select and condense. The author has not selected his subjects carefully enough, nor does he treat them in a masterly manner. Only one-quarter of this volume, for instance, is devoted to the highly important subject of the composition and properties of commercial iron, one-tenth to the sources of supply in nature, and over one-half to the discussion of fuel, coal, coke, producer gas, and the principles of combustion.

Inspecting the details, we find much imperfect and some erroneous information in the discussion of the properties of iron; the subject of oxygen and nitrogen in iron and the causes of blowholes in steel suggests a resumé of current knowledge twenty-five years ago. The description of ore deposits is in parts very faulty and misleading, as, for instance, the statements that the Mesabi ores are "free from phosphorus and sulphur," and that ores of similar quality are found "in Alabama, on the Atlantic Ocean." In the discussion of fuels and combustion, sensible heat of hot gases is called "latent heat" (page 197), and other loose expressions are observable which are not "strictly scientific."

We cannot, in conclusion, praise the book; it contains poorly selected material unskilfully condensed. The three volumes together will be perhaps three-quarters as large as Ledebur's "Eisenhüttenkunde," and probably about one-quarter as good, judging from Volume I as a sample. Let us hope that this extrapolation in judgment will prove wrong.

* * *

Engineering as a Vocation. By **Ernest McCullough,** C.E. Size, 5½ in. x 8 in. (14 cm x 20 cm), 201 pages. Price, $1.00. New York: David Williams Company.

This book is a mixture of advice to young men intending to enter engineering, of information for the benefit of parents, and of some good and some poor ideas on engineers and engineering. The eight chapters deal with the engineer and his work, his education in schools, his self-education; "how to hunt and hold a job" and finally with the worthless discussion as to whether it pays to study engineering.

In discussing the education of the engineer the author states that splendid opportunities exist for men who have missed their chance early in life to secure instruction in correspondence schools and other similar institutions; yet in the very next sentence one reads that young men who can afford the time to study engineering "in the proper manner" have no place in these schools. It is a very serious matter to advise young men to get engineering education in second-rate schools, for if the engineering profession is to be improved, and if there are to be fewer structures falling down in the future, the results will have to be accomplished only by men having thorough training in the fundamentals. The author apparently thinks that our leading technical schools are doing good work and that they constitute the proper means for study. If this be the case, why recommend poor substitutes?

It is difficult to agree with the loose statements that "bridge engineering is practically a distinct profession" and "structural engineering is also a distinct profession." One might as well say that physicians who treat the eye constitute one profession and those who treat the ear another, which is absurd. Certainly bridge and structural engineering are not distinct professions; simply divisions of a large one.

In the first chapter, which is largely devoted to defining the engineer, the two following separate statements appear, indicating the carelessness with which the book has been prepared: "The modern engineer must have a college training or something that is equivalent." "If anything is a fault with the training given in the schools it is that many schools have paid entirely too much attention to outside criticism and the students are narrowly trained specialists, who have been cheated in their unfortunate attempt to get a proper education."

On the other hand, there are some things in the book which would be profitable reading for young men. Advice to acquire industrious habits in college because of the effect on after life is good, and many young men in colleges today would do well to act upon it.

After reading the book one doubts whether the author does or does not believe that engineering is a good vocation to follow and whether or not preliminary training should be taken in the technical schools. The book rather leads one to conclude that some absurd views regarding education recently emanating from Chicago have been somewhat contagious.